TECHNOLOGY QUARTERLY

AND

PROCEEDINGS OF THE SOCIETY OF ARTS.

VOL. XVI DECEMBER, 1903 No. 4

PROCEEDINGS OF THE SOCIETY OF ARTS.

FORTY-FIRST YEAR, 1902–1903.

THE 581st regular meeting of the SOCIETY OF ARTS was held in Room 22, Walker Building, on Thursday evening, April 23, at eight o'clock, Professor Clifford in the chair. Two hundred and fifty persons were present.

After the minutes of the previous meeting had been read and approved, Mr. Francis E. Stanley addressed the Society on "Some Problems to be Solved in the Building of the Perfect Steam-propelled Automobile."

After a brief discussion a vote of thanks was tendered the speaker, and the meeting adjourned.

The 41st Annual Meeting (582d regular meeting) of the SOCIETY OF ARTS was held in Room 22, Walker Building, on Friday evening, May 22, at eight o'clock, Professor Niles in the chair. Two hundred persons were present.

After the minutes of the previous meeting had been read and

approved, Mr. Henry E. Warren was elected to Associate Membership in the Society.

The Annual Report of the Executive Committee was read and approved.

ANNUAL REPORT OF THE EXECUTIVE COMMITTEE PRESENTED AT THE FORTY-FIRST ANNUAL MEETING OF THE SOCIETY OF ARTS, MAY 22, 1903.

The first meeting of the SOCIETY OF ARTS for the present year was held on October 9, 1902. Fourteen meetings have been held, with an average attendance of one hundred and eighty-four.

During the past year the unusual interest in the meetings of the Society which has been shown in the past few years has continued. An effort has been made to have the addresses given before the Society cover a wide range of subjects. As a consequence, nearly all the students in the Institute have had opportunity to hear men who are authorities upon the subjects in which they are preparing themselves for the future.

The following papers have been read:—

"Long Distance Electric Railroading." Professor Louis Duncan, Director of the Department of Electrical Engineering of the Institute.

"The Identification of Our Woody Plants in Winter." Mr. John G. Jack, of the Arnold Arboretum.

"The Art of Lithography." Mr. George H. Bartlett, Principal of the Massachusetts Normal Art School.

"The Designing of Small Racing Yachts." Mr. W. Starling Burgess.

"A Substance with Remarkable Optical Properties, and a Screen Transparent to Ultra Violet Light Only." Professor R. W. Wood, of Johns Hopkins University.

"The Coast Survey and its Work." Mr. O. H. Tittmann, Superintendent of the United States Coast and Geodetic Survey.

"The Reclamation of the Arid Public Lands." Mr. F. H. Newell, Chief Engineer, United States Geological Survey.

"Martinique and Mt. Pelée." Mr. George Kennan.

"Exhibition of Some New Apparatus for Illustrating Certain Electro-Magnetic Phenomena." Professor Louis Derr, of the Institute.

"Government Architecture." Mr. J. Knox Taylor, Supervising Architect, Treasury Department.

"The Composition of Sewage with Relation of Problems of Disposal." Mr. George W. Fuller, Sanitary Expert, New York City.

"The Charles River Dam." Mr. R. H. Dana.

"Some Problems to be Solved in the Building of the Perfect Steam-propelled Automobile." Mr. Francis E. Stanley.

"A Reconnaissance in the Rocky Mountains of British Columbia." Mr. Howard W. DuBois, Mining Engineer, Philadelphia.

At the beginning of the year the Associate Membership was three hundred and sixty-seven. Of these members, three have died and eight have resigned, while six new members have been elected, making the present membership three hundred and sixty-two. There are thirty-seven Life Members.

The TECHNOLOGY QUARTERLY has appeared regularly during the year under the editorship of Dr. Bigelow, with the advice and counsel of the Publication Committee. There have appeared in all twenty-eight articles besides the Proceedings of the Society. A new feature introduced this year is a series of Book Reviews, which, it is hoped, will

add interest to the QUARTERLY, and will further extend the influence of the Institute and members of the Instructing Staff. We are also indebted for valuable contributions to several of our Alumni, namely, Gerard H. Matthes, William Lincoln Smith, G. C. Whipple, and Earle B. Phelps.

GEORGE W. BLODGETT,
EDMUND H. HEWINS,
CHARLES T. MAIN,
DESMOND FITZGERALD,
JAMES P. MUNROE,
} *Executive Committee.*

The report of the Nominating Committee was read, and the following named gentlemen were elected officers of the Society for the year 1903–1904. Executive Committee: George W. Blodgett, Edmund H. Hewins, Charles T. Main, James P. Munroe, and A. Lawrence Rotch. Secretary: James F. Norris.

Mr. Howard W. DuBois then addressed the Society on "A Reconnaissance for the Platinum Metals in the Rockies of British Columbia." The lecture was illustrated with a large number of lantern slides, which had been colored by a new method devised by Mr. DuBois.

A vote of thanks was tendered the speaker, and the meeting adjourned.

J. F. NORRIS, *Secretary.*

ORE DRESSING, OR THE MECHANICAL PREPARATION OF ORES.[1]

By ROBERT H. RICHARDS.

When the miner hoists his ore to the surface, the contained metal may be either in the native uncombined state or combined with other substances, forming minerals of more or less complex composition. In both cases the valuable mineral is always associated with minerals of no value. The province of the ore dresser is to separate the "values" from the waste by mechanical means, obtaining thereby "concentrates" and "tailings." The province of the metallurgist is to extract the pure metal from the concentrates by chemical means, with or without the aid of heat. There are also a number of non-metallic minerals which do not have any value, or at best do not reach their highest value until they have been subjected to some form of mechanical preparation; among them are diamonds, graphite, corundum, garnet, asbestos, and coal. Ore dressing, for the purposes of this article, may be divided into three parts: (1) Properties of minerals which render aid in their separation. (2) Simple operations. (3) Operations combined to form processes or "mills."

1. **Properties.** — The *specific gravity* of minerals varies greatly, some being heavy, others light. The rate of settling in water is affected by the specific gravity in this way: of two particles of the same size but different specific gravity, the heavier settles more rapidly than the lighter; while of two particles of different specific gravity which settle at the same rate in water, that of higher specific gravity is of smaller diameter than the other. The same statements are true in regard to settling in air, and in regard to momentum in air when the particles are thrown out in a horizontal direction. *Color*, *lustre*, and *fracture* are of especial value in hand-picking to aid the eye in selecting the mineral sought. Instances are: of colors, the white of quartz,

[1] Reprinted by permission of the publishers from the article, "Ore Dressing," by Prof. R. H. Richards, in the new volumes of the *Encyclopædia Britannica*. Copyright 1902. The Encyclopædia Britannica Company.

the pale straw color of feldspar, the dull yellow of limonite, the brass yellow of chalcopyrite, the pale metallic yellow of pyrites; of lustres, the vitreous of quartz, the adamantine of diamond and cerussite, the resinous of blende, the earthy of limonite, and the metallic of pyrite; and of fractures, the cleavage planes of feldspar and galena, the conchoidal fracture of quartz and pyrite, the granular of some forms of magnetite and blende. *Magnetism* is a most direct and simple method of separating minerals where it is available. The discovery that, by the use of electro-magnets of great power, minerals formerly regarded as non-magnetic are attracted has made it possible to separate several classes of minerals present in an ore; for example, the strongly magnetic mineral may first be taken out, then the mildly magnetic, and last the weakly magnetic, the non-magnetic being left behind. *Adhesion* acts when brightly burnished particles of gold, issuing with the sand from the stamp mill, come in contact with an amalgamated copper plate, for they are instantly plated with mercury and adhere to the copper, while the sand is carried forward by the water. In this way a very perfect separation of the gold from the sand is effected. In the South African diamond fields it has been found that, if the diamond-bearing sand is taken in a stream of water over a smooth surface covered with a suitable coating of grease, the diamonds will adhere to the grease while the sand does not. *Decrepitation* is the property possessed by some minerals of flying to pieces when heated suddenly. If, for example, blende and barite occur together in a product where the grains are about the same size, they cannot be separated by water, being of nearly the same specific gravity, but when dropped upon a heated iron plate the barite flies into small particles, and can then be separated from the blende by sifting. This principle, however, can only rarely be applied.

2. **Simple Operations.** — The concentration of ores always proceeds by steps or stages. Thus the ore must be crushed before the minerals can be separated, and certain preliminary steps, such as sizing and classifying, must precede the final operations which produce the finished concentrates. The more important of these simple operations will now be described.

The ore as mined contains the valuable minerals attached to and enclosed in lumps of waste rock. The province of *crushing*, or disintegrating, is to sever or unlock the values from the waste, so that the methods of separation are then able to part the one from the other.

In crushing ores it is found wise to progress by stages, coarse crushing being done best by one class of machine, medium by another, and fine by a third. Coarse crushing is accomplished by *breakers* of the Blake type (Figure 1) or of the Gates-Comet type (Figure 2). All of these machines break by direct pressure caused by a movable jaw (*a*, Figures 1 and 2) approaching towards and receding from a fixed jaw (*b*). The largest size ever fed to a breaker is 24 inches in diameter, and the smallest size to which the finest crushing commonly done by these machines brings the ore is about $\frac{3}{4}$ inch in diameter. The machine is generally supplied with ore in lumps not larger than 9 inches in diameter, and crushes them to about $1\frac{1}{2}$ inches in diameter. Medium-

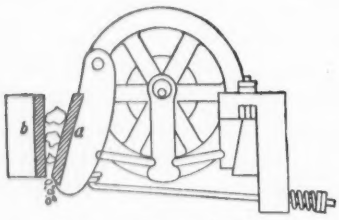

FIG. 1.—BLAKE BREAKER.
a, movable jaw; *b*, fixed jaw.

size crushing is done mostly by rolls (Figure 3) or steam stamps. *Rolls* (Figure 3) crush by direct pressure caused by the ore being drawn between two revolving rolls held closely to-gether. They make the least fine slimes, or *fines*, to be lost in the subse-quent treatment, and are therefore preferred for all brittle minerals.

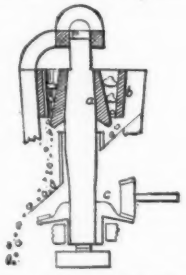

FIG 2.—GATES BREAKER.
a, movable jaw; *b*, fixed jaw; *c*, gear with eccentric hub and with loose fit on the spindle.

The *steam stamp* works upon the same principle as a steam hammer, the pestle being forced down by steam pressure acting through piston and cylinder with great crushing force in the mortar. Steam stamps have been very successful with native copper rock because they break up the little leaves, flakes, and filaments of copper, and render them susceptible of concentration, which rolls do not. Fine crushing is done by gravity stamps, by pneumatic stamps, by centrifugal roller mills, by amalgamating pans, by ball mills, by Chile edgestone mills, and by arrastras.

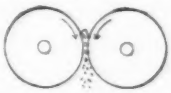

FIG. 3.—CRUSHING ROLLS.

The *gravity stamp* (Figure 4) is a pestle of 800 pounds weight, more or less, which is lifted by a revolving cam, and falls by the force of gravity to strike a heavy blow in the mortar and do the work of crushing; the frequent revolution of the cam gives a more or less rapid succession

of blows. Gravity stamps are especially adapted to the fine crushing of gold ores, which they reduce to $\frac{1}{30}$ inch, and sometimes even to $\frac{1}{50}$ inch grains. The blow of the stamp upon the fragments of quartz not only liberates the fine particles of gold, but brightens them so that they are quickly caught upon the amalgamated plates. The pneumatic stamp has a heavy pestle, which is moved up and down by direct connection with a crank; but since the height of the ore in the mortar varies, the pestle must have the means also of varying its movement to avoid breakage. To this end an air cylinder with piston is inserted in the stem, the elasticity of the air within the cylinder not only giving the needed variation of throw, but increasing the force of the blow in the mortar. The *centrifugal roller mills* are suited to fine crushing of middle products, namely, by-products composed of grains containing both values and waste, since they avoid making such fine slimes. They crush by the action of a roller, rolling on the inside of a steel ring, both having vertical axes. The *amalgamating pan* is suitable for grinding silver ores for amalgamation where the finest grinding is sought, together with the chemical action from the contact with iron. It crushes by a true grinding action of one surface sliding upon another. The *Chile edgestone mill* is employed for the finest grinding ever used preparatory to concentration. The *arrastra*, or drag-stone mill, grinds still finer for amalgamating.

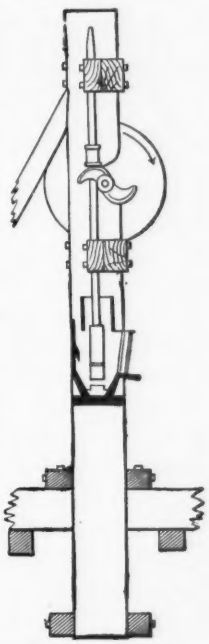

FIG 4. — GRAVITY STAMP.

A considerable class of workable minerals, among which are surface ores of iron and surface phosphates, contain worthless clay mixed with the valuable material, the removal of which is accomplished by the *log washer*. This is a disintegrator, consisting of a long, narrow cylinder revolving in a trough which is nearly horizontal. Upon the cylinder are knives or paddles set at an angle, which serve the double purpose of bruising and disintegrating the clay, and of conveying the cleaned lump ore to be discharged at the upper end of the trough, the water meanwhile washing away the clay at the lower end.

Roasting for Friability. — When two minerals — for example,

pyrites and cassiterite — one of which is decomposed and rendered porous and friable by heat and oxygen, are roasted in a furnace, the pyrites become porous oxide of iron, while the cassiterite is not changed. A gentle crushing and washing operation will then break and float away the lighter iron oxide, leaving the cassiterite behind.

Sizing. — If crushed ore be sifted upon a screen with holes of definite size, two products will result: the *oversize*, which is unable to pass through the screen, and the *undersize*, which does pass. If the latter size be sifted upon another screen with smaller holes, it will again make oversize and undersize. Rittinger adopted for close sizing the following diameters, in millimeters, for the holes in a set of screens: 64, 45.2, 32, 22.6, 16, 11.3, 8, 5.6, 4, 2.8, 2, 1.4, 1. Each of these holes has an area double that of the one next below it; this may be called the screen ratio. A process which does not need such close sizing might use every alternate screen of the above set, and in extreme cases even every fourth screen. In mills the screen ratio for coarse sizes often differs from that for fine. Sizing is done by cylindrical screens revolving upon their axes (Figure 5), by flat, shaking screens, and by fixed screens with a comparatively steep slope. Wire cloth with square holes and steel plate punched with round holes are both used. To remove the largest lumps in the preliminary sizing, fixed bar screens are preferred on account of their strength and durability. Sizes smaller than can be satisfactorily handled by screens are treated by means of *hydraulic classifiers* and *box classifiers* (*Spitzkasten*). The lower limit of screening and, therefore, the beginning of this work varies from grains of 5 millimeters to grains of 1 millimeter in diameter. An hydraulic classifier (Figure 6) is a trough-like washer through which the water and sand flow from one end to the other. In the bottom, at regular intervals, are pockets, or pits, with hydraulic devices which hinder the outflowing discharge of sand (*b*) by an inflowing stream of water (*a*). By regulating the speed of these currents of water, the size of the grains in the several discharges can be regulated, the first being the coarsest, the overflow at the end the finest. Box classifiers are

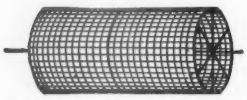

FIG. 5. — TROMMEL, OR REVOLVING SCREEN.

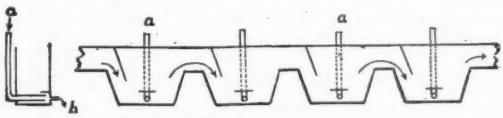

FIG. 6. — HYDRAULIC CLASSIFIER.

similar, except that the pockets are much larger and no inflowing clear water is used; they, therefore, do their work much less perfectly. Classifiers do not truly size the ore, but merely class together grains which have equal settling power. In any given product, except the first, the grain of high will always be smaller than that of low specific gravity. The box classifiers are suited to treating finer sizes than the hydraulic classifiers, and therefore follow them in the mill treatment.

Picking floors are areas on which men, boys, or girls pick out valuable mineral which is rich enough to ship at once to the smelter. The picking is often accompanied and aided by breaking with a hammer. *Picking tables* are generally so constructed that the pickers can sit still and have the ore pass before them on a moving surface, such as a revolving circular table or traveling belt. Stationary picking tables require the ore to be wheeled to and dumped in front of the pickers. Picking out the values by hand has the double advantage that it saves the power and time of crushing, and prevents the formation of a good deal of fine slimes which are difficult to save.

Jigs treat ores ranging from $1\frac{1}{2}$ inches in diameter down to $\frac{1}{50}$ inch. If an intermittently pulsating current of water is passed up through a horizontal sieve on which is a bed of ore, the heavy mineral and the quartz quickly form layers, the former beneath the latter. The machine by which this is done is called a jig (Figure 7), and the operation is called jigging. The pulsating current is obtained by placing a vertical longitudinal partition (*c*) extending part of the way down to the bottom of the jig box. The sieve (*a*) is firmly fastened on one side of the partition, and on the other a piston (*d*) is moved rapidly up and down by an eccentric, causing an up-and-down current of water through the sieve (*a*). The sieve is fed at one end (*e*) with a constant supply of water and ore, and the quartz overflows at the other. The constantly accumulating bed of concentrates is either discharged through the sieve into the space below (*g*), called the hutch, or by some special device at the side. On jigs, where the concentrates pass through the sieve, a bed of heavy mineral grains, too large to pass, holds back the lighter quartz. The quartz overflow from one sieve (*a*) generally carries too much value to be thrown away, and it is, therefore, jigged again upon a second sieve (*b*). In

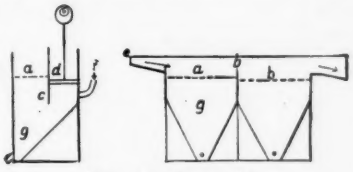

FIG. 7.— HARTZ JIG.

jigging difficult ores, three, four, five, and even six sieves are used. A succession of sieves gives a set of products graded both in kind and in richness, the heavier mineral, as galena, coming first, the lighter, as pyrites and blende, coming later. The best jigging is done upon closely sized products, using a large amount of under-water added beneath the sieve. Very good jigging may, however, be done upon the products of hydraulic classifiers, where the heavy mineral is in small grains and the quartz is large, by using a bed on the sieve and diminished under-water, which increases the suction, or downward pull by the returning plunger.

Bumping Tables. — Rittinger's table is a rectangular, gently sloping plane surface, which by a bumping motion throws the heavy particles to one side while the current of water washes down the quartz to another, a wedge-shaped divider separating and guiding the concentrates and tailings into their respective hoppers. The capacity on pulp of $\frac{1}{30}$ to $\frac{1}{50}$ inch size is some four tons in twenty-four hours. In the Wilfley table (Figure 8), and those derived from it, a gentler vanning motion is substituted for the harsh bump; they have a greatly increased width and a set of riffle blocks (*b*) at right angles to the direction of flow (*c*), tapering in height towards the side where the concentrates are discharged (*d*). This combination has produced a table of great efficiency and capacity for treating grains from $\frac{1}{8}$ inch in diameter down to $\frac{1}{200}$ inch, or even finer. The capacity on $\frac{1}{30}$-inch pulp is from 15 to 25 tons in twenty-four hours.

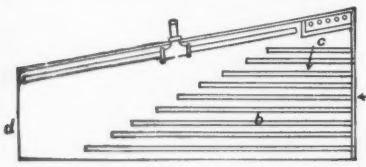

FIG. 8. — WILFLEY TABLE.

Vanners are machines which treat ores on endless belts, generally of rubber, with flanges on the two sides. The belt (Figure 9) travels up a gentle slope (*a*) on horizontal transverse rollers, and is shaken about two hundred times a minute, either sidewise or endwise, to the extent of about one inch. The lower 10 feet is called the concentrating plane (*b*), and slopes 2.78 per cent., more or less, from the horizontal; the upper 2 feet of length is called the cleaning plane (*c*), and slopes 4.45 per cent., more or less. The ore is fed on with water at the intersection of the two planes (*d*). The vibra-

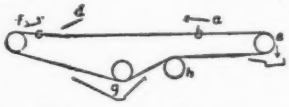

FIG. 9. — FRUE VANNER.

tion separates the ore into layers, the heavy beneath and the light above. The downward flow of the water carries the light waste off and discharges it over the tail roller (*e*) into the waste launder, while the upward travel of the belt carries up the heavy mineral. On the cleaning plane the latter passes under a row of jets (*f*) of clean water, which removes the last of the waste rock. The heavy mineral clings to the belt while it passes over the head roller, and only leaves it when the belt is forced by the dipping roller to dip in the water of the concentrates tank (*g*). The cleaned belt then continues its return journey over the guide roller (*h*) to the tail roller (*e*), which it passes around, and again does concentration duty. Experience proves that for exceedingly fine ores the end-shake, with steep slope and rapid travel, does better work than the side-shake vanner. For ordinary gold stamp-mill pulp, where cleanness of tailings is the most important end, and where to gain it the engineer is willing to throw a little quartz into the heads, the end-shake vanner is again probably a little better than the side-shake, but where cleanness of concentrates is sought the side-shake vanner is the most satisfactory.

Slime tables are circular revolving tables (Figure 10) with flattened conical surfaces and a slope of $1\frac{1}{8}$ inches, more or less, per

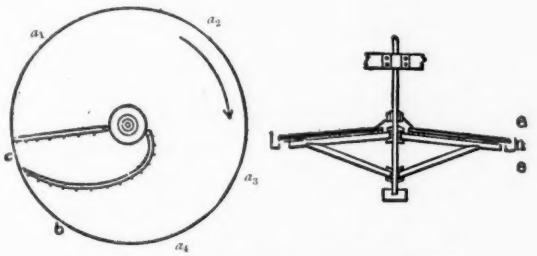

FIG. 10.—CONVEX REVOLVING SLIME TABLE.

foot from center to circumference; a common size is 17 feet in diameter, and a common speed one revolution per minute. These tables treat material of $\frac{1}{100}$ inch and less in diameter coming from box classifiers. The principle on which the table works is that the film of water upon the smooth surface rolls the larger grains (quartz) towards the margin of the table faster than the smaller grains (heavy mineral) which are in the slow-moving bottom current. The material is fed at the centre over one-half the circumference on the side a_1 a_2.

The revolution of the table then discharges the quartz earlier at a_1 a_2 a_3 a_4, an intermediate middling product next at *b*, and the heavy mineral last at *c*. Suitable waste launders and catch boxes are supplied for the three products. The capacity of such a table is 12 tons or more of pulp, dry weight, in twenty-four hours. *Frames* are rectangular slime tables which separate the waste from the concentrates on the same principle as the circular tables, though they run intermittently. They treat very fine pulp, and after being fed for a short period (about fifteen minutes) the pulp is shut off, the concentrates are flushed off with a douche of water and caught in a box, and the feed pulp is again turned on. They are much used in the Cornish tin works. *Canvas tables* are rectangular tables with plane surfaces covered with cotton duck (canvas), free from seams; they slope about $1\frac{1}{2}$ inches to the foot. They are fed with stamp-mill pulp, with the tailings of vanners, or, best of all, with very fine pulp overflowing from a fine classifier. The rough surface of the duck is such an efficient catching surface, that they can run for an hour before the concentrates are removed — an operation which is effected by shutting off the feed pulp, rinsing the surface with a little clean water, and hosing or brooming off the concentrates into a catch box. The feed pulp is then again turned on and the work resumed. They have been more successful than any other machine in treating the finest pulp, especially when their concentrates are finally cleaned on a steep-slope end-shake vanner.

Buddles act in principle like slime tables, but they are stationary, and they allow the sand to build itself up upon the conical surface, which is surrounded by a retaining wall. When charged, the tailings are shoveled from the outer part of the circle, the middlings from the intervening annular part, and the concentrates from the inner part. They treat somewhat coarser sizes than the slime table. When wooden blocks or cobble stones of uniform size are placed in the bottom of a sluice, the spaces between them are called *riffles;* and when gold-bearing gravel is carried through the sluice by a current of water a great many eddies are produced, in which the gold and other heavy minerals settle. The *kieve, or dolly tub*, is a tub as large or larger than an ordinary oil barrel, with sides flaring slightly upwards all the way from the bottom. In the center is a little vertical shaft, with hand crank at the top and stirring blades like those of a propeller at the bottom. Fine concentrates from buddles or slime tables are

still further enriched by treatment in the kieve. The kieve is filled perhaps half full of water, and the paddles set in motion; concentrates are now shoveled in until it is nearly full, the rotation is continued a little longer, and then the shaft is quickly withdrawn and the side of the kieve steadily thumped by a bumping bar as long as settling continues. When this is completed, the water is syphoned off, the top sand skimmed off and sent back to the buddle, and the enriched bottom material shoveled out and sent to the smelter.

3. **Combined Operations.** — In designing concentration works, the millwright seeks so to combine the various methods of coarse and fine crushing, and of preliminary and final concentration, that he may obtain the maximum return from the ore with the minimum cost. Some of the more important of these mill schemes will now be described.

The hand-jig process used for the zinc and lead ores of Missouri is first to clean the ore from adhering clay by raking it back and forth in a sluice with a running stream of water, and then shoveling it upon a sloping screen with holes of about one inch, where it yields oversize and undersize. The former is hand-picked into lead ore, zinc ore, and waste, while the latter is jigged upon a hand jig, and yields several layers of minerals removed by a hand skimmer. The top skimmings are waste, the middle skimmings come back with the next charge to be jigged over, and the bottom skimmings go to a second jig with finer screen. The coarsest of the hutch product goes to the second jig; the finest is sold to a sludge mill. The second jig makes top skimmings, which are sent back to the first jig; middle skimmings, which are zinc concentrates; and bottom skimmings and hutch, which are both lead concentrates. In *the Missouri zinc-concentrating mill* the ore carrying blende and calamine, with a little galena, is in very large crystallization, and contains, when crushed, very little in the way of included grains. It is crushed by Blake breaker and rolls to pass through a sieve with holes $\frac{3}{4}$ inch in diameter, and is then treated on a jig with six consecutive sieves, yielding discharge and hutch products from each sieve, and tailings to waste. The earlier discharges are finished products, while the later are re-crushed and then re-treated on the same jig. The hutch products are treated on a finishing jig with five sieves, and yield galena from the first discharge, and hutch and zinc ore from the others. The capacity of such jigs is very large, even to 75 or 100 tons per day of ten hours.

In *the diamond washing* of Kimberley, South Africa, the material taken from the mine is weathered by exposure to the air and rain for several months, and the softening and disintegration thus well started are completed by stirring in vats with water. Breaker and rolls were tried in order to hasten the process, but the larger diamonds were broken and ruined thereby. The material from the vats is screened and jigged, and of the jig concentrates containing about 2 per cent. of diamonds the coarser are hand-picked and the finer are treated on a greased surface.

Lead and copper ores contain their values in brittle minerals, and are concentrated in mills which vary somewhat according to local conditions; the one here outlined is typical of the class. The ore is crushed by breaker and rolls, and separated into a series of products diminishing in size by a set of screens, hydraulic classifiers, and box classifiers. All the products of screens and hydraulic classifiers are jigged on separate jigs, yielding concentrates, middlings, and tailings; those of the box classifier are treated on the slime table, vanner, or Wilfley table, yielding concentrates and tailings, and perhaps middlings. The coarser middlings contain values attached to grains of quartz, and are therefore sent back to be re-crushed and re-treated. The finer middlings contain values difficult to save only on account of their shape — for example, flat grains — and are sent back to the same machine or to another to be finished.

The *native copper rock of Lake Superior* is broken by powerful breakers, sometimes preceded by a heavy drop hammer weighing nearly a ton. The operation is accompanied by hand-picking, yielding rich nuggets with perhaps 75 per cent. of copper ready for the smelter; at some mines a second grade is also picked out, which goes to a steam finishing hammer, and yields cleaned mass copper for the smelter and rich stamp stuff. The run of rock which passes by the hand pickers is of a size that will pass through bars 3 inches apart, and goes to the steam stamps. The stamps crush the rock to pass through a screen with round holes $\frac{3}{16}$ inch in diameter. This sand is treated in hydraulic classifiers with four pockets, the products from the pockets, being jigged by four roughing jigs yielding finished mineral copper for the smelter, included grains for the grinder, partially concentrated products for the finishing jigs, and tailings which go to waste. The overflow of the hydraulic classifier runs to a tank, the overflow of which is sent to waste in order to diminish the quantity of water, while the discharge

from beneath, treated upon slime tables, yields concentrates, middlings, and tailings. The middlings are re-treated. The stamp mortars are cleaned out periodically, even several times a day, and yield rich nuggets of copper and also unfinished material, which is sent back to the mill. Putting all the finished concentrates together, they will assay from 60 to 80 per cent. of copper according to circumstances. The extraction from the rock is from 50 to 80 per cent. of the copper contained in it.

Cornwall Tin. — Tinstone in Cornwall occurs associated with sulphides, wolfram, quartz, feldspar, slate, etc., and is broken by spalling hammers to 3-inch lumps. Hammers make less slimes than the rock breakers, and they also break the ore more advantageously for the hand-picking. The latter rejects waste, removes as far as possible the hurtful wolfram, and classes the values into groups according to richness. Gravity or pneumatic stamps then crush the ore to $\frac{1}{50}$ inch, and it is then treated on stripes (a species of long, rectangular buddle), which yield heads, middlings, tailings, and fine slimes; the first three are sent separately to circular buddles and the last to frames. The buddles yield heads, middlings, and tailings; the middlings are re-treated, the tailings are all waste; the heads are still further enriched by kieves, which yield tops to the buddle again and bottoms to be shipped to the smelter. The fine slimes are treated on frames, the concentrates of which go to buddles; of these the heads go to kieves.

The *Missouri zinc-lead sludge mill* takes the finest part of the hutch product of the hand jigs. The treatment begins on revolving screens with two sizes of holes — 25 millimeters and 1 millimeter; these take out two coarser sizes, of which the coarsest is waste and the other is jigged, yielding concentrates and waste. The main treatment begins with the finest size, which is much the largest product. It is fed to a convex circular buddle (first buddle), and yields a coarser product at the outer and a finer product in the inner part of the circle. The finer product is treated by a series of buddlings which vary somewhat, but in general are as follows: fed to a second buddle, it yields zinc and lead ore in the centre, next zinc ore, then middlings, which come back, and, outside of all, tailings. The zinc-lead ore is set to one side until enough has accumulated to make a buddle run, when it is run upon a third buddle, yielding in the central part pure lead concentrates, next lead ore (which is returned to this treatment), next zinc ore, and, outside of all, a zinc product, which is fed to the second

buddle. The coarse outside product of the first buddle is treated in much the same way as the fine, but it yields practically no lead-zinc product, which simplifies the series of buddlings necessary.

Gold Mill. — Gold ores usually contain their value in two conditions — the free gold, which can be taken out by mercury, and the combined gold, in which the metal is either coated with or combined with compounds of sulphur, tellurium, etc. The usual gold-milling scheme is to crush the ore by rock breaker to about $1\frac{1}{2}$ inches in diameter, and then to crush with water by gravity stamps, a little mercury being added to the mortar from time to time to begin the amalgamation at the first moment the gold is liberated. The pulp leaves the mortar through a screen with holes or slots $\frac{1}{30}$ to $\frac{1}{50}$ inch in width, and is then passed over amalgamated plates of copper or silver-plated copper. The free gold, amalgamated by the mercury, adheres to the mercurial surface on the plate; the rest of the pulp flows on through mercury traps set to catch any of the mercury which drains off the end of the plate. The plates and mortar are periodically cleaned up, the plates being scraped to recover the amalgam and leave them in good condition to do their work; if plates are used inside the mortar they are cleaned in the same way. The residue of partly crushed ore in the mortar, with amalgam and free mercury scattered through it, is ground for a time in a ball mill, panned to recover the amalgam, and returned to the mortar. The pulp flowing away from the mercury traps flows to a Frue vanner, or Wilfley table, on which it yields concentrates for the chlorination plant or the smelter and tailings; the latter are waste when the heavy mineral is of low grade, but if the vanner concentrates are of high grade they still contain values in very fine sizes, which can and should be saved. Improved methods for saving this material have been made recently in California. The vanner tailings are sent to a fine classifier, from which the light overflow only is saved; this is treated upon canvas tables, yielding concentrates and tailings, and these concentrates, treated upon a little end-shake vanner with steep slope and rapid travel, give clean, very fine, high grade concentrates for the chlorination works.

Iron Ore. — The brown ores of iron from surface deposits are contaminated with a considerable amount of clay and some quartz. The crude ore from surface pits or shallow underground workings is treated in a log washer, and yields the fine clay, which runs to waste, and the coarse material, which is caught upon a screen and hand-picked, to free

it from the little quartz, or jigged if it contains too much quartz. The magnetic oxide of iron occurs associated with feldspar and quartz, and can often be separated from them by the magnet. The ore, after being broken by breaker and rolls to a size varying from $\frac{1}{8}$ to $\frac{1}{16}$ of an inch in diameter, goes to a magnetic machine, which yields (1) the strongly magnetic, (2) the weakly magnetic, and (3) the non-magnetic portions. The second, or middlings, product contains grains of magnetite attached to quartz, and is, therefore, re-crushed and sent back to the magnets; the strongly magnetic portion is shipped to the furnace, and the waste to the dump heap. In concentrating certain zinc sulphides, siderite (carbonate of iron) follows the zinc and would seriously injure the furnace work. By a carefully adjusted roasting of the product in a furnace, the siderite is converted into magnetic oxide of iron, and can then be separated from the zinc ore. A special magnet of very high power has been designed for treating the franklinite of New Jersey — a mineral which is non-magnetic in the usual machines. The ore, crushed by breaker and rolls and hand-picked to remove garnet, is treated upon a belt with a roughing magnet to take out the most magnetic portion, and is then very closely sized by screens with 16, 24, 30, and 50 meshes per linear inch, respectively. The several products are fed to separate magnetic machines, yielding the franklinite, which is treated by the zinc-oxide grates, and afterward by the spiegel furnace; the residue, which is jigged, and yields the zinc silicate and oxide for the smelter furnaces and waste carrying the calcite, quartz, and mica.

Gold dredging is the method employed for saving the values from gold-bearing sand and gravel in river bottoms. A flat boat is built, provided with guy ropes or spuds, or both, for holding it still. The spuds are vertical posts, or legs, which can be forced down by powerful tackles through sockets in the boat, and so lift it, as it were, upon legs. At one end of the boat is a powerful endless-chain bucket-dredge, which scrapes the gravel from the bottom and elevates it to a revolving screen in the boat. This, in turn, sifts out the bowlders, which are at once elevated to the bank of the river, while the fine material flows over tables covered with cocoa matting, which acts like fine riffles, catching the gold in the interstices. The matting is periodically lifted up and thoroughly rinsed off. The rinsings are panned for gold, and the matting returned for another charge. The fine sand and gravel passing over the end of the tables is sent by the elevator to the river bank.

Asbestos, when of good quality, is in compact masses, which by suitable bruising and beating are resolved into fine, flexible fibers. The Canadian asbestos is associated with serpentine. It is crushed by breakers to $\frac{3}{4}$ inch, and then screened on $\frac{1}{16}$-inch screens to reject fines. The values are removed by hand-picking, and are crushed by rolls carefully set so as not to break the fiber. This product is then sized by screens, and the various sizes are sent to the Cyclone pulverizer, which by beating liberates the individual fibers. It then goes to a screen with eleven holes to the linear inch, and yields a granular undersize and oversize, and also a fibrous oversize which is drawn off by a suction fan to a settling chamber with air outlets covered by fine screen cloth. This fibrous product is the clean mineral for the market. A special treatment separates the fibers of different lengths.

The usual method of dressing *corundum and emery*, after the preliminary breaking, is to treat the material in an edgestone mill fitted with light wooden rollers. The action is that of grinding one particle against another, whereby the talc, chlorite, mica, etc., are worn off from the harder mineral. A constant current of water carries off the light impurities. This is called the "muller" process. At Corundum Hill, North Carolina, the first step in removing the impurities from "sand" corundum is to subject it to the scouring action of a stream of water while it is being sluiced from the mine to the mill, the action being increased by several vertical drops of 5 to 10 feet in the sluice. After reaching the mill, all that will not pass through a 14-mesh screen is crushed by rolls, and the undersize of the screen is treated in a washing trough; this removes part of the light waste, and the "mullers" mentioned above complete the cleaning.

Graphite occurs in schist, but being of less specific gravity than the other minerals which enter into the composition of the schist, it settles later than they do. It also breaks into thin scales, which reduces its settling rate still further. The ore is broken by breakers and by Chile edgestone mills or by gravity stamps to a size varying with the character of the minerals, the size varying from perhaps $\frac{1}{20}$ to $\frac{1}{60}$ inch in diameter. The pulp is then conveyed through a series of settling tanks, of which the later are larger than the earlier. The quartz and other waste minerals settle in the earlier tanks, while the graphite settles later; the latest tank gives the best graphite. In the Dixon Company's works in New York some forms of concentrators are believed to have replaced the slower settling tanks.

Gold Hydraulicking. — The gold contained in superficial gravel deposits is saved by one of the several schemes included in hydraulicking. A source of water power with very high head furnishes the water to a powerful jet called a "giant"; this washes down the gravel bank, and conveys the coarse and fine material first to a head sluice with riffles, which may be 100 feet or more long, and then to a tail sluice, also with riffles, which may be several miles long. Periodically — once a month, perhaps — the head sluice is cleaned up, the riffle blocks being removed and cleaned, and the sand which is washed down and caught in a temporary tank at the foot panned for its gold with hand pans. The riffle blocks are then replaced and work proceeds as before. At the end of the season the long tail sluice also is cleaned up in the same way. Gravel containing five cents a cubic yard has in places paid well.

The *phosphates* of Florida are of four kinds: hard rock, soft rock, land pebble, and river pebble. The hard rock is crushed by toothed rolls and cleaned in log washers. The washed product is screened; the sizes finer than $\frac{1}{16}$ inch are thrown away because too poor; the other sizes are dried and sold, some waste having been picked out of the coarsest. The soft rock is simply dried, ground, and sold. Land pebble is treated by log washers, any clay balls remaining being removed by a screen, and the phosphate dried and sold. In special cases land pebble is treated by hydraulicking, followed by a log washer, and this again by a powerful jet washer, to remove the last of the clay. River pebble is taken from the river by centrifugal pumps, and screened on two screens with 1-inch and $\frac{1}{16}$-inch holes, respectively; the oversize of the first sieve and the undersize of the second sieve are thrown away because of too low grade.

SCHOOL LABORATORY WORK: A FREE-MILLING GOLD RUN.[1]

By ROBERT H. RICHARDS and E. E. BUGBEE.

Friends often ask how accurate the work of our little stamp mill is, or express the opinion that a little mill cannot do good work. As a reply we would like to place on record the results of a few tests.

There are two periods of work in our laboratory: the fall term, when the class collectively goes through all our chief laboratory processes; and the spring term, when the students work up their theses, each electing some one process or investigation, and devoting himself wholly, or almost wholly, to that.

Some of the tests, the results of which are recorded in this paper, were made in the fall term; the preparation of the apparatus, the sampling, and the assaying being done largely by assistants, with the aid of the students. Others were made in the spring term as thesis work, and the results are due wholly to the students.

The complete "tree" of our process is patterned after the most approved California mill practice. It is modified only where the small-scale method must differ from the large, in order to secure results from a brief test lasting a few hours, followed, perhaps, by an interval of many months between tests, which seek to rank in accuracy with those of the large-scale work, running over months and years of continuous work, broken only by brief intervals for cleaning up.

We use no battery plates, because we believe the loss due to absorption of gold by the copper may be greater than the gain from using them.

We use silver amalgam on our apron plate, painting it on before the run and scraping it off with the gold contents afterwards, for two reasons: it renders the plate very quick to catch gold, and it renders the loss from absorption by the copper plate insignificant, or prevents it altogether.

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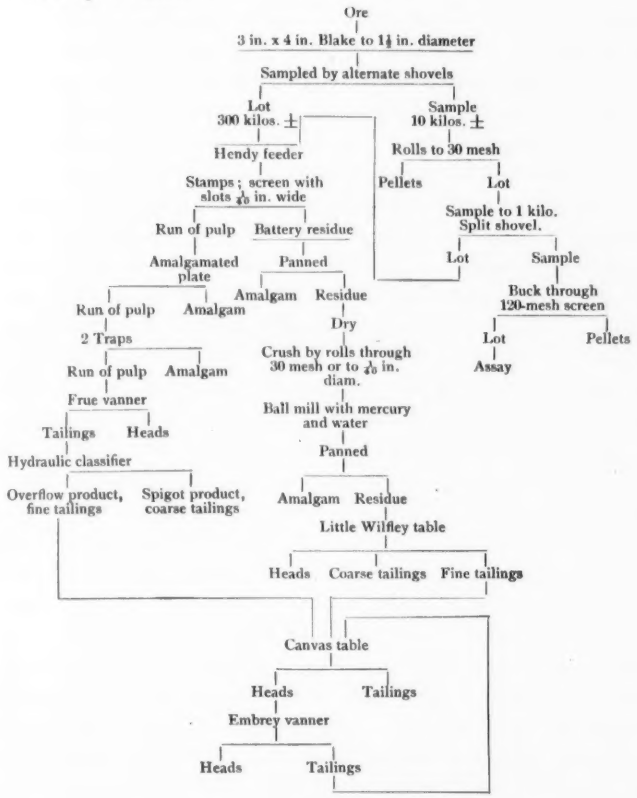
[1] Reprinted from the *Transactions of the American Institute of Mining Engineers*, Albany meeting, February, 1903.

Tree of Process.



Our canvas and fine vanner plant for treating the extremely fine tailings is that known as the Gates canvas plant, originally designed by G. G. Gates and installed at the Kennedy mine, Jackson, California. Since it has been demonstrated clearly that this canvas plant can pay only when the assay of concentrates is high or when the percentage of concentrates is large, we do not always choose this run as the one to illustrate to students the wonderful catching power of canvas for extremely fine concentrates. We do, however, make it our practice to separate, by classifying, the coarse Frue-vanner tailings from the fine, thereby preparing for canvas should we decide to use it.

Sampling. — Many of the ores we run would come under the head of "rich spotty" ores, as the Colorado phrase is. To crush and cut

down such an ore, according to best Colorado rules of sampling, would require breaking the whole batch of from 500 to 1,000 pounds down to 10 or even 30 mesh. This would ruin our gold stamp-mill run. We are thus put in the odd position of having to choose between sampling the lot according to the best rules for safe work and spoiling the stamp run, or of making a stamp-mill run, using only a very crude sample for valuation, which we could not possibly defend on the score of accuracy. It is needless to say we choose the latter method.

The mill and vanner were run under the following conditions:

Apron plate of copper 6 feet long, 2 feet wide, covered with silver amalgam; number of stamps, 3; weight of each stamp, 225 pounds; diameter of shoe, 5.75 inches; number of drops, 98 per minute; height of drop, from 4.25 to 5.75 inches; height of discharge, 4.75 inches; screen, horizontally slotted, openings $\frac{1}{40}$ inch wide; rate of crushing, 57 to 94 kilos. per hour; plate slope, 1.5 inches per foot; feed water to the battery, 15 kilos. per minute; wash water on vanner, from 4 to 8 kilos. per minute; Frue vanner slope, 3.5 to 4.5 inches in 12 feet; number of shakes, 198 per minute; belt travel, from 40 to 60 inches per minute; canvas table, 10 feet long, 4 feet wide; canvas-table slope, 1.5 inches per foot; canvas-table feed, 10 kilos. water per minute and from 0.7 to 0.8 kilos. dry ore per minute; Embrey vanner, 1.5 inches per foot slope, 80 inches per minute travel of belt, 240 vibrations of 0.5 inch per minute.

The mill is a little 3-stamp Fraser and Chalmers prospecting mill of 1882. It is very satisfactory in all respects except the number of stamps. A school battery should have either two or five stamps; the former is preferred by us. Three stamps cannot have an order of drop which satisfactorily distributes the sand.

The vanner is a standard 4-foot Frue, made by Fraser and Chalmers in 1882. It is a very good one, the belt being today as good as new. It is run from five to ten days in a year.

The canvas table is supplied with an accurate distributer. The canvas is No. 6 cotton duck, with the warp laid across the table for the greatest roughness, and is cut from a roll woven 10 feet wide. The feed water is the quantity, and the pulp is two to three times the quantity used by the Gates canvas plant. The copper plate is $\frac{1}{8}$ inch thick, of softest copper.

The amalgam is made by dissolving 95 grammes of silver in 380 c.c. of nitric acid (1.2 sp. gr.), with addition of 817 c.c. of water after solu-

tion. Pure mercury is then added at the rate of 16 parts of mercury to 1 part of silver. It should be stirred at short intervals during the formation of the amalgam. The result is a soft amalgam, clean and free from crystals or mercurous nitrate. Failure to follow this rule exactly is liable to give a very impure amalgam, which will require time and trouble to clean.

The apron plate is cleaned preparatory to the application of silver amalgam as follows: it is washed off with water, scoured with wet infusorial earth or very fine sand, and then washed off with water again. Obstinate black spots can be removed by scouring with a flat piece of pumice stone and water, adding mercury to plate the cleaned copper. It is then washed all over with a strong solution of cyanide, adding mercury, again rinsed off with water, and then rubbed dry with clean cloth or cotton waste which must be free from oil. A moderate quantity of mercury is now rubbed on without water, after which the plate is ready for the amalgam. No water is allowed upon it yet.

The clean amalgam is squeezed in chamois skin to a moderately hard ball, and rubbed without water on the clean, bright, amalgamated copper surface of the apron plate. (A flat, bristle paste brush may be useful to obtain an even distribution.) This coating is quick to catch gold, and prevents the gold from coming in contact with the copper. No water is allowed upon it until stamping begins.

After the run, the plate is rinsed off with water, and the amalgam scraped off as clean as possible with a rubber scraper. More mercury is rubbed on, and the plate is again scraped. This is repeated twice more, and all the amalgam is put together. If there is any hard amalgam which has not been removed, it is gently rubbed with a flat piece of pumice stone in the presence of water and mercury, and so removed without injury to the plate. The amalgam from the plate, combined with that from the traps, is retorted; the residue is melted with borax in a crucible, and the bullion is parted for gold.

The battery residue is worked up, as shown in the tree of the process, yielding amalgam, Wilfley heads, coarse tailings, and fine tailings, which are added to the main portions of these products.

Since the battery is fed with pure mercury, while the apron plate is coated with silver amalgam, the battery amalgam enables us to get a valuation on the fineness of the gold caught at this point. We have no means to get this ratio on the portion of the gold caught on the apron plate.

The mercurous nitrate and a little residual silver nitrate from making silver amalgam are worked over for mercury and silver by immersing strips of iron in the liquid, which precipitates the metals as amalgam.

The results of seven tests are given here.

ORE NO. 2,423, FROM NOVA SCOTIA, NOVEMBER 12, 1901. CLASS WORK.

	WEIGHT, KILOS.	ASSAY, OZ. PER TON.	GOLD, PER CENT.	GOLD, GRAMMES.	PER CENT. OF THE TOTAL GOLD IN CRUDE ORE.
Ore fed	384.8	1.03	0.003531	13.5873	. .
Products:					
Battery and ball-mill gold	. .	. .	. . .	14.1545	87.40
Plate and trap gold	. .	. .	. . .	0.6786	4.19
Vanner and Wilfley heads	19.9	0.84	0.002868	0.5708	3.53
Vanner and Wilfley coarse tailings	225.2	0.06	0.000213	0.4794	2.96
Vanner and Wilfley fine tailings	139.0	0.06	0.000206	0.2860	1.77
Unaccounted for	0.7	*1.03	0.003531	0.0247	0.15
Total	384.8	. .	. . .	16.1940	100.00

* Assumed for purposes of calculation.

Per cent. of gold contents saved by amalgamation . . .	91.59
Per cent. of gold contents in vanner concentrates . . .	3.53
Total	95.12
	Oz. per ton.
Assay by fire	1.03
Assay based on mill run	1.23

ORE NO. 2,425, FROM NOVA SCOTIA, NOVEMBER 19, 1901. CLASS WORK.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed	266.1	0.06	0.0002057	0.5475	. .
Products:					
Battery and ball-mill gold	. .	. .	. . .	0.03966	5.40
Plate and trap gold	. .	. .	. . .	0.27700	37.72
Vanner and Wilfley heads	11.6	0.06	0.0001936	0.02246	3.06
Vanner and Wilfley coarse tailings	178.9	0.05	0.0001695	0.30326	41.34
Vanner and Wilfley fine tailings	62.1	0.03	0.0001628	0.06387	8.70
Unaccounted for	13.5	*0.06	0.0002057	0.02777	3.78
Total	266.1	. .	. . .	0.73402	100.00

* Assumed for purposes of calculation.

Per cent. of gold contents saved by amalgamation . . .	43.12
Per cent. of gold contents in vanner concentrates . . .	3.06
Total	46.18

	Oz. per ton.
Assay by fire	0.06
Assay based on mill run	0.08

ORE NO. 2,444, FROM NOVA SCOTIA, NOVEMBER 4, 1902. CLASS WORK.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed	298.2	0.16	0.0005486	1.636	. .
Products:					
Battery and ball-mill gold	. .	. .	. . .	2.0300	78.43
Plate and trap gold	. .	. .	. . .	0.1481	5.72
Vanner and Wilfley heads	13.8	0.42	0.001440	0.1987	7.68
Vanner and Wilfley coarse tailings	186.7	0.02	0.00006857	0.1280	4.94
Vanner and Wilfley fine tailings	81.6	0.025	0.00008570	0.0699	2.70
Unaccounted for	16.1	*0.025	0.00008570	0.0138	0.53
Total	298.2	. .	. . .	2.5885	100.00

* Assumed for purposes of calculation.

Per cent. of gold contents saved by amalgamation . . .	84.15
Per cent. of gold contents in vanner concentrates . . .	7.68
Total	91.83

	Oz. per ton.
Assay by fire	0.16
Assay based on mill run	0.253

ORE NO. 2,444, FROM NOVA SCOTIA, NOVEMBER 11, 1902. CLASS WORK.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed	338.8	0.15	0.0005142	1.742	. .
Products:					
Battery and ball-mill gold	. .	. .	. . .	1.9969	75.16
Plate and trap gold	. .	. .	. . .	0.2022	7.61
Vanner and Wilfley heads	15.0	0.42	0.001440	0.2160	8.12
Vanner and Wilfley coarse tailings	205.2	0.02	0.00006857	0.1407	5.29
Vanner and Wilfley fine tailings	107.4	0.025	0.00008570	0.0921	3.46
Unaccounted for	11.2	*0.025	0.00008570	0.0096	0.36
Total	338.8	. .	. . .	2.6575	100.00

* Assumed for purposes of calculation.

Per cent. of gold contents saved by amalgamation . . .	82.77
Per cent. of gold contents in vanner concentrates . . .	8.12
Total	90.89
	Oz. per ton.
Assay by fire	0.15
Assay based on mill run	0.229

ORE NO. 2,444, FROM NOVA SCOTIA, NOVEMBER 18, 1902. CLASS WORK.

	WEIGHT, KILOS.	ASSAY, OZ. PER TON.	GOLD, PER CENT.	GOLD, GRAMMES.	PER CENT. OF THE TOTAL GOLD IN CRUDE ORE.
Ore fed	332.6	0.25	0.0008570	2.850	. .
Products:					
Battery and ball-mill gold	. .	. .	. . .	1.9820	76.44
Plate and trap gold	. .	. .	. . .	0.0850	3.28
Vanner and Wilfley heads	12.9	0.40	0.001373	0.1769	6.82
Vanner and Wilfley coarse tailings	201.2	0.03	0.0001028	0.2069	7.98
Vanner and Wilfley fine tailings	114.1	0.035	0.0001200	0.1370	5.28
Unaccounted for	4.4	*0.035	0.0001200	0.0053	0.20
Total	332.6	. .	. . .	2.5931	100.00

* Assumed for purposes of calculation.

Per cent. of gold contents saved by amalgamation . . .	79.72
Per cent. of gold contents in vanner concentrates . . .	6.82
Total	86.54
	Oz. per ton
Assay by fire	0.25
Assay based on mill run	0.227

ORE NO. 2,444, FROM NOVA SCOTIA, FEBRUARY, 1902. THESIS OF C. H. BURR AND B. E. MCKECHNIE.

	WEIGHT, KILOS.	ASSAY, OZ. PER TON.	GOLD, PER CENT.	GOLD, GRAMMES.	PER CENT. OF THE TOTAL GOLD IN CRUDE ORE.
Ore fed	356.2	0.13	. . .	. .	. .
Products:					
Battery gold	. .	. .	. . .	2.01902	72.53
Plate gold	. .	. .	. . .	0.22736	8.17
Vanner heads	16.8	0.44	0.001511	0.25380	9.39
Vanner coarse tailings	208.6	0.02	0.00007401	0.15444	5.54
Vanner fine tailings	122.3	0.03	0.00009900	0.12107	4.35
Unaccounted for	8.5	*0.02	0.00007	0.00566	0.20
Total	356.2	. .	. . .	2.78135	100.18

* Assay assumed for purposes of calculation.

Per cent. of gold contents saved by amalgamation . . .	80.70
Per cent. of gold contents in vanner concentrates . . .	9.39
Total	90.09

	Oz. per ton.
Assay of ore by fire	0.13
Assay based on mill run	0.222

ORE NO. 2,361, FROM NOVA SCOTIA, MARCH, 1901. THESIS OF HENRY C. MARCUS.

	Weight, Kilos.	Assay, Oz. per Ton.	Gold, Per Cent.	Gold, Grammes.	Per Cent. of the Total Gold in Crude Ore.
Ore fed	256.30	0.13	0.000444	1.13797	. .
Products:					
Battery gold	. .	. .	. . .	2.38216	74.84
Plate gold	. .	. .	. . .	0.51440	16.16
Vanner heads	15.65	0.11	0.000377	0.05904	1.86
Vanner coarse tailings	195.85	0.02	0.000069	0.13156	4.13
Vanner fine tailings	43.45	0.057	0.000195	0.08493	2.67
Unaccounted for	5.35	*0.057	0.000195	0.01043	0.33
Total	256.30	. .	. . .	3.18252	99.99

* Assumed for purposes of calculation.

Per cent. of gold contents saved by amalgamation . . .	91.00
Per cent. of gold contents in vanner heads	1.86
Total	92.86

	Oz. per ton.
Assay of ore by fire	0.13
Assay based on mill run	0.36

SUMMARY OF THE END RESULTS OF THE SEVEN TESTS.

Ore Number.	Percentage of Gold in Crude Ore.						Assay by Fire, Oz. per Ton.	Assay by Mill Test, Oz. per Ton.
	Battery and ball mill.	Plate and trap.	Vanner and Wilfley heads.	Vanner and Wilfley coarse tailings.	Vanner and Wilfley fine tailings.	Unaccounted for.		
2,423	87.40	4.19	3.53	2.96	1.77	0.15	1.03	1.23
2,425	5.40	37.72	3.06	41.34	8.70	3.78	0.06	0.08
2,444	78.43	5.72	7.68	4.94	2.70	0.53	0.16	0.253
2,444	75.16	7.61	8.12	5.29	3.46	0.36	0.15	0.229
2,444	76.44	3.28	6.82	7.98	5.28	0.20	0.25	0.227
2,444	72.53	8.17	9.39	5.54	4.35	0.20	0.13	0.222
2,361	74.84	16.16	1.86	4.13	2.67	0.33	0.13	0.36

Attention is called to the slight variation in the mill assay of the four tests on No. 2,444, and also to the considerable variation in the assays by fire.

Remarks. — The method here used in tabulating the results of a test has been found to be a most concise and satisfactory one. As the value of the ore obtained by the mill run is much more accurate than that based on the assay of the ore, the percentages found in the column headed "Per cent. of the total gold" are based on the total gold in the ore as found by the mill run, and not on the assay of the feed; in fact, one of the objects of this work is to show the student the impossibility of valuing a free-milling gold ore by sampling and assaying.

The degree of accuracy attained here is, we believe, up to that of the very best mill practice. It has been attained by locating the causes of error, and then by removing them as completely as possible, even by departing, if necessary, from large-scale mill methods.

THE WATER SUPPLIES OF SOUTHEASTERN ALASKA.

By ELLEN H. RICHARDS.

Two considerations led me to take the portable laboratory [1] with me on the long journey: first, that its use might be tested under very different conditions from those under which the Mexican water supplies were examined;[2] and second, because a record of the condition of the available waters in this early stage of the development of the country must have a certain value in the future.

The tables may seem to be meagre in comparison with those which are made from the testing of gallon samples in a fully equipped laboratory, but it is believed that a good idea may be obtained of the general character of the waters on the given date.

At present there are no "wells," as the term is generally applied, in the coast towns visited. In all the localities where there is anything approaching level ground there has accumulated from the luxuriant vegetation rich black peaty soil of great depth; a thickness of 5 to 15 feet is common.

In the rainy season this holds water like a sponge, so that hollows scooped out 4 or 5 feet deep, with sides supported by a rough wooden curb or left sloping inwards, readily fill with water, which collects again as it is dipped out.

These pools serve the Indians for the scanty supply they use, and as they are dug on the hillside at some distance from the houses, which are huddled together on the shore, there does not seem great danger from contamination other than from unclean vessels and hands used in dipping up the water.

The 25th of July was toward the end of the dry season, and the pools were dry or very low, although the vegetation was rank and vigorous.

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[1] Technology Quarterly, Vol. XIV, p. 295.

[2] *Transactions*, A. I. M. E., Mexican meeting, Nov., 1901.

This black water is quite a feature of the country. In the vicinity of Sitka, for instance, where it has collected in lakelets as black as ink; indeed, containing iron and tannin, it is one variety of ink. The white population do not use this water (one of the ship's officers said, "The water is poison all along this coast"), but depend very largely upon cisterns, which in that climate seem to furnish abundance of good water. Large tanks having a capacity of 200 barrels and more are frequently found, and furnish water for baths and laundry purposes in small places like Fort Wrangell, where the little stream brought from the mountains for the use of the sawmill is highly charged with iron.

At the Indian village of Killisnoo, situated on an island where the fish-oil factory is in operation only part of the year, instead of collecting the rainfall, the engineer of the factory brings water in a scow from a creek on the mainland, and allows the inhabitants to use it from the tap in the works, otherwise they depend on the dug pits. Judging from the all-pervading odor, it is probable that cistern water would absorb the odor from the factory to a decided degree. This water was the only sample obtained on the trip which gave high results in chlorine and ammonia.

Most of the towns are able to utilize mountain streams; sometimes these are really of glacial water, which many persons consider dangerous for drinking.

At Juneau some further provision must be made if it is to realize its ambition and become the capital of Alaska, since the use of the streams for hydraulic work is liable to interfere with the present good quality. Even now it runs so turbid that it is filtered through a bank of sand before entering the pipes.

On Douglas Island, opposite Juneau, $1,000,000 have been spent in cutting a trench fifteen miles long, intercepting springs and streams in order to obtain a sufficient supply for the Alaska-Treadwell mills for eight months in the year. Therefore this should be a typical sample.

At Ketchican, as at many places in the far north, the only supply in winter comes from melted snow.

Sitka has a heavy rainfall, 60 to 95 inches annually, and is well supplied with cisterns, large and small. The sample Number 17 in the table was taken from the large cistern of the United States Agricultural Experiment Station, which furnishes the house with hot and cold water. The station occupies the site of the old Castle Baranoff, on an eminence nearly surrounded by water.

Indian River is a stream of good volume, clear and cold, running in a shallow layer over pebbles about half a mile back from the town, from which drinking water for a large portion of the inhabitants is hauled in barrels, or in the same square oil cans used in Mexico. It was delicious to the taste. This stream was said to have "mineral" in it, and was preferred to the cistern water. The analysis does not seem to bear out this statement. It may be that in times of high water some taste is perceived, but the mineral content is low. This stream could be utilized for a town supply, if the demand was sufficient for the installation of water works.

The nearest to an ideal supply and the most intelligent use of it is at Skagway, where a lake of considerable area — Lake Dewey — gives an abundant quantity of excellent quality. A reservoir was built in 1898 in a depression in the mountain cliff far above the town, where the water is stored, and from which it is distributed under the high pressure due to the elevation of the reservoir.

It was gratifying to find a considerable knowledge of the sources and character of the various waters even among the common people. This may be because so many of them are from the Puget Sound region, where remarkable intelligence is shown in securing safe water. Seattle and Portland are in strong contrast with San Francisco and many cities in the Middle West, as well as in the East, where it is difficult to secure appropriations for water works. At Missoula, Montana, for instance, we were told by a waitress that the town supply came from Rattlesnake Creek, was stored in a reservoir, and that it was the purest in the state. Many more highly educated people in the East are ignorant of the source and quality of the drinking water.

Typhoid fever is almost unknown in Seattle, which obtains water from Cedar River, thirty miles from the city. The water is stored in reservoirs, and is called by the citizens "the finest in the country."

A question of great interest is, does the rule of normal chlorine hold on the Pacific coast as on the Atlantic and in Mexico?

In order to give an approximate answer, samples were carefully collected from typical stations and taken to Berkeley, California, where the determinations were made through the courtesy of the authorities of the university. My thanks are due to Professors Hilgard, Jaffa, and Colby for the facilities so kindly placed at my disposal.

In so far as conclusions may be reached by the results on single samples, it would seem that we may rely on limits previously set,

Analysis of Alaskan Waters.

(Parts in 1,000,000.)

Number.	Locality.	Date of		Physical.			Total Residue on Evaporation.	Nitrogen as			Hardness.	Chlorine.	Iron.
		Collection.	Examination.	Color (Massachusetts standard).	Turbidity.	Sediment.		Free ammonia.	Nitrites.	Nitrates.			
1	Ketchican town supply	1903 July 23	1903 July 24	.05	0	0	. .	0	0	0	0.0	1.95	. .
2	Ketchican cistern	"	"	.50	Very slight	0	. .	.050	0	0	16.9	1.77	. .
3	Fort Wrangel Pool	24	"	1.0	0	0	. .	.030	0	0	16.9	. .	000
4	Fort Wrangel large cistern	"	"	1.8	Slight	0	. .	.010	0	0	11.1	1.16	. .
5	Fort Wrangel supply from hills to sawmill and town	"	"	1.5	0	0	. .	.030	0	0	1.6	1.23	.200
6	A sizable stream at Tonka cannery	"	"	0.2	0	0	. .	0	0	0	14.3	0.80	.020
7	Juneau town supply	"	25	0.01	0	0	47.8	0	0	0	62.9	0.85	. .
8	Douglas Island, Alaska-Treadwell supply	28	28	.10	Slight	Slight	41.2	0	0	.010	11.5	1.39	000
9	Lake Dewey, supply of Skagway, 1898 Res.	25	26	.08	0	0	. .	0	0	0	14.3	0.92	. .
10	White Pass R.R., large falls	"	"	. .	. .	. .	. .	. .	. .	. .	. .	0.59	. .
11	White Pass R.R., small stream from mountains	"	"	.02	0	0	. .	0	0	0	7.9	. .	. .
12	Tank in White Pass Station from stream near boundary	"	"	.03	Slight	0	. .	.030	0	0	7.9	. .	. .
13	Head waters of Yukon River	"	"	.10	Distinct	Slight	. .	.030	0	0	1.6	. .	. .
14	Stream from Davidson Glacier	"	"	Too turbid	Distinctly milky	Clayey	. .	0	0	0	11.5	0.40	. .
15	Killisnoo	26	27	0.7	Slight	Slight	. .	.700	.005	.050	112.0	Very heavy Pr.	
16	Indian River, Sitka	27	"	0.0	None	Very slight	21.4	.010	0	0	14.3	3.3	. .
17	Cistern, Agricultural Station, Sitka	"	"	0.3	Slight	Slight	. .	.040	.001	.080	8.3	3.9	. .

namely, that beyond 75 to 100 miles from a rocky shore the normal chlorine never exceeds 1.00 part per million, and on the other hand that it is never entirely absent, falling to .4 or .6 only.

A glance at the map will show that Juneau and Skagway are just about one hundred miles from the open ocean, that there is no breaking of the waves on rocks such as to drive spray high in the air nearer than fifty miles at White Pass, while Fort Wrangell and Ketchican are more exposed on the south to the open sea in Dixon's Entrance. Douglas Island, surrounded by salt water, gives only 1.39 parts of chlorine, and this, as has been shown, is a typical sample of mixed water approaching a yearly normal.

Davidson Glacier is the eastern portion of the Muir and, therefore, is not exposed to the winds from the Pacific shore of the island, and may be considered to show the residual chlorine in the air sweeping over the entire country, as in our own Adirondacks and western slopes of the White Mountains.

At Sitka, on the other hand, surrounded by the ocean, within a few miles of the dashing waves, the chlorine content of the water is even higher than in the waters of the sand-surrounded Nantucket and Martha's Vineyard. It may be noted here that the waters supplying the cities of Seattle, Spokane, and Walla Walla show 1.50 parts per million, according to Professor Byers, of the State University.

As bearing on the question of the interpretation of the significance of nitrites and nitrates in water, it is to be noted that they are generally absent in the waters from these primeval hills, rich in vegetation as they are. Even the black water from the thick peat failed to yield nitrates, which are so abundant in the black prairie soil of Illinois, once the home of the buffalo. Possibly this may indicate a necessity for inoculation with nitrifying organisms before agriculture will be successful.

The absence of nitrites, even in samples showing ammonia, may also be due to absence of animal contamination.

On the whole the country seems to be well provided with good soft water fairly accessible. It is to be hoped that medical men will make careful record of the effects noticed from the use both of the glacial waters and the peaty waters, so that the question of their harmfulness may be settled.

NOTES ON THE WATER SUPPLIES IN THE BLACK HILLS OF SOUTH DAKOTA AND VICINITY.

By ELLEN H. RICHARDS.

Too often the first settlers, whether they are pioneer agriculturists or miners, neglect ordinary precautions as to the purity of the drinking water. Through this neglect the sources become polluted, epidemics follow, and not until some strong pressure is put upon them do either class apply the necessary remedy. When a mining company has lost a few of its most valued men by fever it begins to think it more economical to bring in safe water, even at considerable expense. The company is, to its credit, usually generous to the community when once it has introduced the good water. There are discouragements in changing the habits of a frontier town if once they have depended on wells. The ordinary workman does not take much stock in sanitary theories; because the glass of water does not take effect as quickly as the bullet, he scoffs at danger, and his family fall victims to his ignorance. Criminal indifference occurs higher in the scale. At one hotel, kept with decided carefulness in most respects, a certain spring water was advertised to be exclusively used on the table. One day the cart failed to go to the spring, and the town supply, dangerous in quality, was substituted, apparently unnoticed by most of the guests.

Education all along the line is needed, although the railroad companies take much better care of the water supplied to the dining cars than was the case a few years since.

No better work can be undertaken by any state board of health than a sanitary survey of the waters of the state, the results to be published widely. The addition to information of a geological significance would be considerable; for instance, in Spearfish cañon the layers of white rock are called gypsum, but none of the waters in that valley show sulphates.

In the semi-arid regions this examination is increasingly important, especially because a study of peculiarities is needed to help in the interpretation of the results both before and after the use of water for

irrigation purposes. If this knowledge is not obtained, some serious results may follow the re-use of irrigation water.

For instance, in the case of the spring water brought to Edgemont (see table), what is the significance of the high nitrates? Do they indicate an old buffalo range, or was this spring a favorite resort for the Indians? Or is there a cause in present surroundings?

The contemporaneous presence of ammonia and nitrites would seem to indicate the latter, and to demand an examination of conditions, even though a condemnation of this source would mean bringing in by train water from several score of miles.

An instructive set of results is given by the samples from Deadwood and Lead. The stream, coming originally from the hills, has been used for several mining and milling operations, and the presence of ammonia and nitrites indicates a degree of contamination. The precipitous and rocky bed gives no chance for the formation of nitrates in quantity; the constant addition of organic matter reduces them if they are formed. The typhoid rate is high among those who do not boil the water. The cemetery of the town is situated on the one approximately level spot, on a cliff of some 400 feet elevation above the town, and is a favorite resort on account of the view. From a crevice in this cliff about 50 feet above the street there issues a copious stream of clear, cold water, much prized by the inhabitants. It is named White Rock in the table. This shows the greatest previous pollution of any sample secured. The special spring in the town of Lead also indicates a similar pollution.

A further confirmation of the value of nitrates as an indication is seen in the set from Sylvan Lake, which is in granite rock in the region of Harney's Peak. It is soft surface water, confined within its walls only three years ago, and resembles our Eastern waters. The clear, cold spring near it is evidently beginning to feel the presence of the cottages and the stable above it, while the stream at the watering place is the typical water of the South Black Hills, like that which supplies the town of Custer. The well used by the railroad did not give so good an account of itself, showing nitrites and chlorine. An investigation revealed large cattle yards and several houses near by.

Another interesting sample is the Giant Spring at Great Falls, Montana. This large body of water boils up in the very edge of the Missouri River. Its source is unknown, but must be many miles distant. It is a hard water, but shows no nitrates, although sulphates

SOUTH DAKOTA WATERS.

(Parts in 1,000,000.)

LOCALITY.	DATE OF		PHYSICAL.			NITROGEN AS			SO_3.	HARDNESS.	CHLORINE.
	Collection.	Examination.	Color (Massachusetts standard).	Turbidity.	Sediment.	Free ammonia.	Nitrites.	Nitrates.			
	1903	1903									
Kidney Spring, Deadwood	June 19	June 20	0	0	0	0	0	0	Considerable	196.0	Faint
White Rock Spring, Deadwood	"	"	0	0	0	0	0	7.000	Distinct	260.7	Heavy
Spring from other side of White Rock Hill, Deadwood	"	"	0	0	0	0	0	.400	Slight	. .	Slight
Stream. Town supply, Deadwood	"	"	Dark turbid	Decided	Considerable	.050	.006	0	. .	164.0	No turbidity
Supply, Lead	22	"	0	0	0	. .	0	0	. .	295.0	Very slight
Special spring, Lead	"	"	0	0	0	. .	.002	4.000	. .	. .	Heavy
Little Spearfish River at Falls	23	23	0	0	0	. .	0	0	No turbidity	238.0	No turbidity
Spearfish River at town	"	"	0	0	0	. .	0	0	"	245.0	"
Fountain in hotel at Spearfish from spring	"	"	0	0	0	. .	0	0	"	260.7	"
Sylvan Lake	24	24	.1	0	Slight	.050	0	0	"	24.7	"
Spring under rock near entrance to Sylvan Lake	"	"	0	0	0	.010	0	.200	"	22.1	Slight
Watering place on the road from Custer	"	"	0	0	0	0	0	0	"	14.3	No turbidity
Burlington and Missouri R.R. wells at Edgemont*	25	25	0	0	Slight	.300	.040	.400	Heavy	148.4	Heavy
Spring water hauled from the hills for the town and hotel drinking water	"	"	0	0	0	.150	.025	5.000	Distinct	228.2	Distinct

* Alkali water; cattle drink it with reluctance.

abound and chlorine is present (an examination made in 1898 gave 4.5 parts of chlorine, .100 parts of nitrate, and 316 parts of hardness per million).

In lead ore districts, of which the Cœur d'Alene, Idaho, is an example, the mineral formation complicates the problem, and distilling the drinking water seems the only way to be certain of its safety, if for no other reason, because the excessive amount of iron is objectionable.

Spokane is fortunate in one way in having Cœur d'Alene Lake as its reservoir, but since the lake is already well seeded with Anabœna, it is sure to have trouble sooner or later; indeed, these soft waters, if stored in the bright sunlight in the midst of the dusty plains, are very inviting feeding grounds for many water plants. One of the coming problems is, then, the exclusion of all such as give odors and tastes from the very inception of the works.

NOTES ON THE METALLURGY OF COPPER OF MONTANA.[1]

By H. O. HOFMAN.

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[1] Reprinted from *Transactions of the American Institute of Mining Engineers*, Albany meeting, February, 1903.

I. Introductory.

In 1901[1] this country produced 268,782 long tons of fine copper. Of this total, Montana is credited with 102,621 long tons, or 38.2 per cent.

The leading smelting works are located in Butte, Anaconda, and Great Falls. In Butte are the works of the Colorado Smelting and Mining Company, the Colusa Smelter of the Montana Copper Company (closed in 1893), the plant of the Parrot Silver and Copper Company (closed in 1899), the old Bell plant (closed in 1884), the Clark's Colusa (closed in 1893); then come the active works of the Colusa Parrot Mining and Smelting Company (Butte Reduction Works), the Butte and Boston Consolidated Mining Company, and the Montana Ore-Purchasing Company.

In Anaconda are the old and new (Washoe) works of the Anaconda Copper Mining Company; in Great Falls the establishment of the Boston and Montana Consolidated Copper and Silver Mining Company.

The active works furnished in 1901, approximately, the following amounts:

Anaconda Copper Mining Company	113,800,000 pounds.
Boston and Montana Consolidated Copper and Silver Mining Company	54,700,000 "
Montana Ore-Purchasing Company	29,900,000 "
Butte and Boston Consolidated Mining Company	22,200,000 "
Butte Reduction Works	18,000,000 "
Colorado Smelting and Mining Company	7,500,000 "
	246,100,000 "

The discrepancy of 17,000,000 pounds of copper between the figures of production is due to the fact that the matte of some works is converted in others, and that both are credited with the copper they turned out.

II Condensed Account of Past and Present Plants.

Colorado Smelting and Mining Company. — Started 1879 by the late Henry Williams.[2] The ore was roasted at first in the long-hearth

[1] *United States Geological Survey*, Mineral Resources, 1901, p. 160.

[2] R. W. Raymond, *Engineering and Mining Journal*, July 19, 1902; J. Douglas, *The Mineral Industry*, IV, p. 283; C. W. Goodale, *International Mining Congress*, Butte, 1902.

hand reverberatory furnace, having a hearth 12 by 50 feet, and then smelted in a reverberatory matting furnace, with hearth 14 feet 7 inches by 9 feet, wood being used as fuel. The original smelting furnaces treated 12 tons of ore in twenty-four hours, producing matte assaying 60 per cent. copper and from 700 to 800 ounces silver to the ton, which had to be hauled 200 miles to reach the railroad. In 1889 Brückner cylinders came into use for roasting ores, followed in 1892 by the R. Pearce single-deck turret furnace. In 1894 R. Pearce's first double-deck turret furnace was added, and in 1901 the first six-deck turret furnace of R. F. Pearce was put in operation. At present (summer, 1902) the roasting department has four single-deck turret furnaces with 6-foot hearths, five double-deck turret furnaces (one with a 6-foot and four with a 7-foot hearth), and one six-deck turret furnace with a 7-foot hearth.

The hearths of the reverberatory matting furnaces were made 21 by 12 feet in 1889, increasing the above capacity to 18 tons in twenty-four hours. At present there are three matting reverberatories with hearths 47 by 20 feet (the first of these larger furnaces having been erected in 1899). The matte containing copper from 50 to 55 per cent., silver, 100 ounces, and gold, 1.5 ounces to the ton is sold.

Colusa Smelter of the Montana Copper Company. — Started 1880 by A. Wartenweiler, with four long hand reverberatories for roasting (hearths 60 feet by 11 feet 6 inches), and two reverberatory matting furnaces (hearths 15 feet 9 inches by 10 feet). Smelting begun 1881; a reverberatory matting furnace treated in twenty-four hours 12 tons of raw and roasted ores, producing 65-per cent. copper matte, with a consumption of ten cords of wood. Lump ore was at first heap-roasted, but, stopped on account of the smoke nuisance, was replaced in 1882 by stalls.[1] In 1884[2] there were in operation, besides the stalls, twelve long-hearth hand reverberatory roasting, six reverberatory matting, and one blast furnace. The latter, 3 feet by 4 feet 6 inches at the tuyères and 10 feet high, was run with charcoal. Works sold 1888 to the Boston and Montana Consolidated Copper and Silver Mining Company, and formed the so-called Upper Works. They closed down in 1893 when the new Great Falls establishment of the company made them superfluous.

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[1] *Mineral Resources of the United States*, 1891, p. 94.

[2] *Mineral Resources of the United States*, 1883–84, p. 339.

Parrot Silver and Copper Company.—Started July, 1881. Ore was at first roasted in eleven long hand reverberatories (hearths 60 by 14 feet) and smelted in six matting reverberatories (hearths 14 feet by 11 feet 6 inches). Weight of smelting charge, 2 to 3 tons; 10 tons put through a furnace in twenty-four hours. Matte (60 to 67 per cent. copper and 40 to 45 ounces silver per ton) at first brought forward to blister copper, but this soon given up on account of the excessive cost and the fact that matte was more easily marketed than blister copper. Shortly after starting of works, a stall 40 feet deep and 9 feet high[1] was built by J. T. Reese to roast lump ore. It proved a failure, and was replaced by A. J. Schumacher with open stalls 8 feet deep, 6 feet high, and 6 feet wide.[2] In 1891 there were in operation 120 stalls.[3] The hand reverberatory roasting furnaces were supplemented, 1889, by two small Spence mechanical shelf-roasting furnaces,[4] each treating 5 tons of concentrates in twenty-four hours, reducing sulphur to from 6 to 7 per cent.; these were replaced in 1892 by a round and in 1893 by two oblong Keller-Gaylor-Cole mechanical shelf burners.[5]

A Roberts four-hearth mechanical reverberatory roaster, made by the Western Iron Works, of Butte, and similar to the Wethey furnace,[6] was put in operation 1897. Hearths of matting reverberatories enlarged in 1887 to 16 feet by 12 feet 6 inches. For lack of space they never reached the large sizes of the other furnaces at Butte. The largest furnace, 1895, had a hearth of 22 feet by 16 feet. In 1884 a Herreshoff 48-inch water-jacket blast furnace was erected by E. D. Peters; in 1885 a 3- by 6-foot blast furnace, with cast-iron water jackets, was added, but was soon replaced by A. J. Schumacher with a 3- by 8-foot furnace having steel jackets. In 1884 the converting of copper matte was begun under the general direction of the inventor of the process, Pierre Manhès. The practice was to melt 40- to 50-per cent. copper matte in a cupola furnace, tap it into converter, blow to white metal, and pour; then melt the white metal in the cupola, and blow it to blister copper in the converter. In 1885 A. J. Schumacher

[1] Width unknown.

[2] E. D. Peters, *Mineral Resources of the United States*, 1883–84, p. 389; also *Modern Copper Smelting*, 1895, p. 181.

[3] *Mineral Resources of the United States*, 1891, p. 981.

[4] Peters, *Modern Copper Smelting*, 1895, p. 220.

[5] Peters, *op. cit.*, p. 214; Hofman, *Metallurgy of Lead*, 1899, p. 191.

[6] Hofman, *op. cit.*, 1899, p. 195; *The Mineral Industry*, 1898, VI, p. 449.

succeeded in making metallic copper from 40- to 50- per cent. matte in one operation of two consecutive stages, and thus laid the foundation of the modern practice of converting.

When the works were closed down August 20, 1899, the roasting department contained two Keller-Cole-Gaylord furnaces, each having a capacity of 50 tons ore, and one O'Hara furnace, treating 50 tons in twenty-four hours. Smelting was carried on in three undergrate-blast reverberatory furnaces with hearths 20 by 12 feet, treating altogether 135 tons of ore in twenty-four hours, and in two blast furnaces 44 inches by 96 inches at the tuyères, each putting through daily 120 tons of charge. The converting department had six upright converters 5 feet in diameter and 8 feet 6 inches in total height.

Bell Plant. — Soon after the erection of the Parrot works followed the building of the Bell plant, which roasted in long-hearth reverberatories and smelted in blast furnaces. In 1885[1] it had two hand roasters and two blast furnaces.

Clark's Colusa. — Built 1884 by W. A. Clark; sold 1888 to Boston and Montana Company, and formed its Lower Works; closed 1893 in favor of the new plant at Great Falls. At this plant the first two-hearth mechanical O'Hara roasting furnace was operated, 1885; the hearths were 50 by 7 feet. A second O'Hara furnace, erected 1885,[2] had hearths 70 by 7 feet, and treated 30 tons ore in twenty-four hours. In 1891[3] ores were roasted in three O'Hara furnaces and one 10-ton Brückner cylinder. The first smelting furnace, a circular water-jacket blast furnace, 36 inches in diameter at the tuyères and 9 feet high, was replaced 1885 by a larger furnace. In 1887 two 50-ton circular water-jacket blast furnaces were put in operation. In 1886 two reverberatory matting furnaces were added to the plant, each treating 12 tons of charge in twenty-four hours.

Colusa-Parrot Mining and Smelting Company's Plant (*Butte Reduction Works*). — Started 1885 by employees of the Parrot Smelter. At first ore was roasted in heaps and then smelted (45 tons in twenty-four hours) in a 36-inch water-jacket blast furnace. A long hand reverberatory (hearth 100 by 14 feet), erected 1886, remained in operation until 1888, when it was cut in two, making two furnaces 50 by 14 feet each. In the same year the first Brückner furnace was installed (cylinder 15

[1] *Mineral Resources of the United States*, 1885, p. 216.

[2] *Mineral Resources of the United States*, 1885, p. 216.

[3] *Mineral Resources of the United States*, 1891, p. 94.

by 7 feet, charge 7 tons); in the fall of 1889 a second cylinder, 20 by 8 feet. In 1894 the first four-deck Wethey calciner was built, replacing one of the 50- by 14-foot hand reverberatory roasters and the 15- by 7-foot Brückner cylinder. Its work proving satisfactory, a second Wethey furnace was erected. The 50- by 14-foot hand roaster was replaced by a larger furnace, 75 by 5 feet, to be used mainly for roasting slimes. In 1898 the third and in 1900 the fourth Wethey furnace was installed, replacing the remaining hand roaster and Brückner cylinder. Smelting capacity of the plant increased, 1886, by erection of a 96- by 42-inch Orford brick furnace; the original 36-inch furnace put out of blast and replaced in 1889 by one of 38 inches at tuyères. In 1897 a sectional blast furnace of the United Verde Hot-Air type, 84 by 42 inches at tuyères, replaced the two older furnaces. In 1900 a second rectangular furnace, 112 by 42 inches at tuyères, was put in blast, and in 1901 the 84- by 42-inch furnace was enlarged to 112 by 42 inches.

First reverberatory matting furnace (hearth 15 by 9 feet) erected 1887; the second (hearth 18 by 11 feet), treating from 12 to 15 tons in twenty-four hours, erected 1888. It was then customary to wet down roasted ore, and shovel it into matting furnace through side doors. In 1889 hoppers placed over the furnaces, and roasted ore charged hot through the roof. This increased smelting capacity from 18 to 21 tons in twenty-four hours. In 1896 a large reverberatory matting furnace (hearth 35 by 16 feet, capacity 45 to 50 tons ore in twenty-four hours) replaced the two smaller furnaces. A second (hearth 45 by 16 feet, capacity 55 to 70 tons) was built in 1897; and in 1900 a third (hearth 50 by 20 feet, capacity 85 to 100 tons). This last did so well that the 1896-furnace was rebuilt on the lines of the furnace of 1900. Present plant: four Wethey four-deck calciners, two water-jacketed blast furnaces (112 by 42 inches at tuyères), and three reverberatory matting furnaces (hearths 45 by 16, 50 by 20, and 50 by 20 feet).

Butte and Boston Consolidated Mining Company. — Smelting works started 1885 with four Brückner cylinders (8 by 18 feet), two reverberatory matting furnaces (hearths 22 by 17 feet, treating 5-ton charges), and two blast furnaces, the larger being 96 by 36 inches at the tuyères and 11 feet high. In 1889 two O'Hara roasting furnaces were added; later, four Allen-Brown-O'Hara furnaces; and these giving satisfaction, the original O'Hara's were remodeled to the Allen-Brown-O'Hara type,

and the Brückner cylinders thrown out. In 1892, when the plant burned down, eight Allen-Brown-O'Hara furnaces were in operation. The plant was immediately rebuilt, the Allen-Brown-O'Hara furnaces were repaired, but not the Brückner cylinder. A new blast furnace, 96 by 36 inches and 16 feet high, was erected to take the place of the old small cupola. This furnace was originally provided with a bell-and-hopper feed, which proved too expensive. In 1901 the width of the new furnace was increased to 40 inches, and the height reduced to 11 feet. Up to 1893 the blast furnaces smelted little but roasted concentrates. Matting reverberatory furnaces remodeled and enlarged in 1895. Converter plant started in 1900. Present plant: eight Allen-Brown-O'Hara roasting furnaces (hearths 90 by 9 feet), four matting reverberatories (hearths 50 by 20 feet, the first furnaces of this size having been built in 1899–90), one blast furnace 96 by 44 inches at tuyères and 9 feet high, and three horizontal converters with barrels 10 feet 6 inches by 7 feet.

The Montana Ore-Purchasing Company. — Ground broken for smelting works October 17, 1892; first matte tapped January 1, 1893. Ore roasted in two Allen-Brown-O'Hara furnaces (hearths 9 by 100 feet, each treating 60 tons in twenty-four hours), and smelted in two reverberatory matting furnaces (hearth 12 feet 10 inches by 20 feet, and fire box 5 by 6 feet; natural draught, stack 80 feet high and 4 by 4 feet in the clear, and charge 7 to 8 tons; time of melting three and five-tenths to four hours), and one blast furnace 42 inches diameter at tuyères. After a year's run, blast furnace replaced by two matting reverberatories like those first built. In 1894 two Herreshoff furnaces (9 feet inside diameter, 5 hearths) were added to roasting department. These giving satisfaction, eight more were erected at once, thirty in 1895, and thirty in 1899, making in all seventy Herreshoffs. The O'Hara's destroyed by fire 1901, and not rebuilt. Early in 1894 hearths of matting reverberatories were enlarged to 13 feet 6 inches by 25 feet, fire boxes, however, remaining 5 feet by 6 feet; at same time fuel was changed from lump to slack coal, and undergrate blast substituted for natural draught. These changes increased the tonnage per furnace for twenty-four hours by 10 to 15 tons. Furnaces again enlarged 1898, and two more built 1899, making six in all (hearth 16 by 33 feet, fire box 6 by 8 feet, stack 80 feet high and 5 by 5 feet in the clear).

January, 1900, two blast furnaces (42 by 120 inches at tuyères;

11 feet from tuyères to throat; cast-iron water jackets) were blown in. These water jackets giving much trouble by cracking, furnaces were replaced by larger furnaces (42 by 160 inches at tuyères and 10 feet 6 inches in height) with steel water jackets. Converting plant started August, 1893, with two vessels of the Parrot type (5 feet external diameter); in 1894 two were added, and in 1899–1900, four more. Two barrel converters (bowls 10 feet 6 inches long and 7 feet in diameter), installed 1901, are in constant use, and the upright Parrot converters are worked only when these are unable to handle all the matte. It is the intention to erect two additional horizontal and to remove the remaining upright converters. Present plant: seventy Herreshoff roasters, six reverberatory matting furnaces, four blast furnaces, two horizontal and eight upright converters.

The Anaconda Copper Mining Company. — At Anaconda there are three separate plants, the Upper, the Lower, and the New or Washoe Works. The Upper Works of the Anaconda Copper Mining Company were started by Marcus Daly in 1883 with thirty-four hand reverberatory roasting furnaces (hearths 50 by 14 feet), twenty-six reverberatory matting furnaces (hearths 20 by 14 feet), and two 70-ton water-jacket blast furnaces. A reverberatory smelting charge weighed 3 tons, and 15 tons were treated in a furnace in twenty-four hours. In 1885 two Brückner roasting furnaces were added. After fire of 1889 works rebuilt, and remained practically unchanged until closed down in November, 1901. At that time they contained forty Brückner roasting furnaces with cylinders 18 by 9 feet, four MacDougall roasting furnaces (16 feet diameter and 21 feet high, with six hearths), four Wethey calciners, with hearths 100 by 12 feet, and eleven reverberatory matting furnaces, with hearths 31 by 16 feet, a matting furnace taking 9-ton charges and treating 60 tons of ore in twenty-four hours.

The Lower Works of the Anaconda Copper Mining Company, located about two miles below the Upper Works, were built under Marcus Daly by O. Stallmann. The smelting department was started December, 1888, with fifty-six Brückner cylinders (9 feet 6 inches by 18 feet) and twenty-eight reverberatory matting furnaces, with hearths 22 by 16 feet, a matting furnace receiving 6-ton charges and treating 40 tons ore in twenty-four hours. The smelter buildings, which were entirely of timber, were totally destroyed in March, 1889, by a fire which started in the coal bins. They were quickly replaced by a steel structure. In 1890 forty Brückner furnaces of the same size as the first

ones were added to the plant, and in 1899 were added four MacDougall furnaces of the Evans-Klepetko pattern (16 feet diameter and 21 feet high, with six hearths). Of the twenty-eight original matting furnaces, seven were enlarged in 1899; three had hearths 32 by 18 feet, 33 by 18 feet, and 45 by 18 feet; and four, hearths 35 by 18 feet. They were charged with about 11 tons of ore, and averaged 75 tons in twenty-four hours.

The converter plant was started in 1890 with fifteen stalls for upright vessels, 10 feet high and 6 feet in diameter. In the same building were six blast furnaces 8 feet by 3 feet 6 inches at tuyères and 10 feet high.

New Electrolytic Refining Plant,[1] still in full operation, started in 1894, refines part of the converter copper of the Washoe works, the rest being shipped to Baltimore for treatment. The old experimental plant,[2] started in 1891 and closed in 1893, contained 630 tanks.

The New or Washoe Works[3] of the Amalgamated Copper Company were planned and erected by F. Klepetko. They are located on the site selected by the Washoe Copper Company (incorporated about 1893) for a new plant. When the Amalgamated Copper Company obtained control of the Anaconda and Washoe Companies, it was decided to erect new works instead of remodeling and enlarging those of the Anaconda Copper Company. The works were started in 1902, contain forty-eight six-hearth MacDougall-Evans-Klepetko roasting furnaces 16 feet diameter and 21 feet high; five blast furnaces (56 by 180 inches at the tuyères and 14 feet 10 inches high); fourteen matting reverberatories (hearth 20 feet by 50 feet 2 inches), and eight converter stands (barrels 12 feet 5 inches long and 8 feet in diameter). The fumes from these four departments are drawn off by four steel stacks, 200 feet high and lined throughout (inner diameter 20 feet; outer, 23 feet 6 inches). Since fumes damaged crops in the Deer Lodge Valley, it has been decided to deliver them 500 feet higher up into the air. The fumes from the four departments will be conducted through branch flues into one brick main 2,300 feet long, built on a hillside, and ending in a stack 300 feet high and 30 feet inner diam-

[1] *Berg- und Hüttenmännische Zeitung*, 1893, p. 53. *Zeitschrift für Electrochemie*, 1901, p. 793. *Engineering and Mining Journal*, Sept. 19, 1896.

[2] *Mineral Resources of the United States*, 1891, p. 92.

[3] *Engineering and Mining Journal*, March 1, 1902. *Western Mining World*, March 8, 1902.

eter. The main will be 37 feet deep; the first 1,300 feet will be 60 feet wide, and the remaining 1,000 feet 120 feet wide. Underneath will be a tunnel to receive the cars, into which will be discharged the flue dust. It is expected that the new addition will be finished this summer.

Boston and Montana Consolidated Copper and Silver Mining Company's Great Falls Plant. — Ground broken in the spring of 1890. Plant first consisted of twenty-four Brückner roasters (cylinders 8 by 16 feet), eight tilting reverberatory matting furnaces (hearth 13 by 16 feet), and two 5-ton upright converters. Matting furnaces having regenerative chambers and fired with producer gas resembled the Campbell open-hearth steel furnaces. Plant has been greatly enlarged since 1890, under the management of F. Klepetko, and furnaces much changed to meet constant demand for increased capacity. Present plant: twenty-four Brückner cylinders (8 by 16 feet), all idle except two or three used for drying slimes from the concentrator; eighteen six-hearth MacDougall-Evans-Klepetko roasting furnaces (16 feet diameter and 21 feet high); five gas-heated stationary matting reverberatories with regenerative chambers, four of which have hearths 42 feet 6 inches by 15 feet 9 inches, and one a hearth 29 by 12 feet; five blast furnaces (18 feet high to charging floor; four are 56 by 180 inches, and one 44 by 180 inches at tuyères); twelve 5-ton upright converters (7 feet diameter and 14 feet 7 inches high); one anode-casting reverberatory (hearth 14 feet 8 inches by 22 feet 6 inches) with Walker casting machine; and an electrolytic copper refinery with 312 tanks and four copper-refining furnaces of different sizes.

The early history of this company may be summarized by the beginnings of the several operations: concentrator, March, 1891; Brückner cylinders, April, 1892; reverberatory matting furnaces, April, 1892; converters, August, 1892; blast furnaces, April, 1893; refining furnaces, January, 1893; electrolytic refinery, February, 1893.

In concluding this introductory account, it should be stated that nearly all smelting works have concentrating works in which second-class ore is enriched.

III. The Ores.

The ores of the Butte district are classified by Goodale[1] as copper-silver ores, silver-copper ores, and silver ores. The copper-bearing minerals of the copper-silver ores are chalcocite, bornite, enargite, and chalcopyrite; the gangue is quartz and feldspar. The copper- and silver-bearing minerals of the second class are wurtzite, sphalerite, bornite, chalcopyrite, pyrite, tetrahedrite, enargite, tennantite, and galenite; the gangue is quartz and feldspar, with some barite and fluorite. Of the silver ores, which are of secondary importance, the silver-bearing minerals are pyrite, sphalerite, galenite, tetrahedrite, and argentite; the gangue is quartz, rhodocrosite, and rhodonite. With regard to metallurgical treatment, the copper-bearing ores are divided into first and second class. The line of division varies with the different mines. In general, first-class ore contains 10 to 15 per cent. copper, and is smelted direct, usually in the blast furnaces, without having been roasted. Second-class ore, which forms about 90 per cent. of the ore mined and contains from 3 to 6 per cent. copper, is concentrated; fine concentrates are always roasted and, as a rule, smelted in the reverberatory matting furnaces; coarse concentrates, above 5 millimeters in size, are often added to the blast-furnace charges without having been roasted, the proportion amounting to from 30 to 35 per cent. of the charge.

The following partial analyses of 1902, taken from smelting records, give the general character of the ore now treated. The analyses of 1896, given by Goodale[2] and here quoted for comparison, show much higher values.

[1] *Transactions American Institute of Mining Engineers*, XXVI, p. 599.

[2] *Op. cit.*, p. 639.

TABLE I. — Analyses of Butte Ores.

Plant.		A.												B.	C.			D.	
		First-class ores.	First-class ores.	First-class ores.	First-class ores.	Second-class ores.	Coarse concentrates.	Coarse concentrates.	Fine concentrates.	Fine concentrates.	Fine concentrates.	Slimes.	Slimes.	Average.	First-class ores.	Second-class ores.	Concentrates.	First-class ores.	Concentrates
Cu . . .	21.35	15.15	13.80	8.60	8.05	5.9	15.85	15.60	13.45	10.80	9.50	7.20	4.80	7.00	12.7	5.2	8.7	9-10	10
Fe . . .	10.75	19.50	. .	18.25	12.10	. .	21.95	22.00	25.90	22.60	22.00	8.40	5.10	24.00	18.4	16.0	27.0	16-20	30
Zn . . .	0.50	trace	. .	0.55	. .	. .	0.50	. .	trace	. .	trace	. .	trace	12.00	. .	. .	. .	. .	. .
S	20.30	28.93	. .	23.50	15.37	. .	29.10	29.70	34.70	29.50	28.46	11.70	7.40	32.00	24.8	19.8	34.7	20-25	34-39
Mn . . .	. .	trace	. .	. .	trace	. .	trace	. .	trace	. .	trace	. .	trace	. .	. .	. .	. .	. .	. .
As . . .	1.70	2.10	. .	0.70	1.26	. .	0.95	. .	0.40	. .	0.90	. .	0.10	. .	. .	. .	. .	. .	. .
Sb . . .																			
Pb . . .	trace	trace	. .	trace	trace	. .	trace	. .	trace	. .	trace	. .	trace	. .	. .	. .	. .	. .	. .
Al_2O_3 . .	3.15	7.00	. .	6.15	10.60	. .	3.00	. .	5.70	. .	7.60	. .	20.70	. .	4+	. .	. .	. .	. .
CaO . . .	0.50	0.28	. .	0.30	0.54	. .	0.45	. .	0.60	. .	0.00	. .	trace	. .	. .	. .	. .	. .	. .
MgO . .	trace	trace	. .	trace	trace	. .	trace	. .	trace	. .	trace	. .	trace	. .	. .	. .	. .	. .	. .
Insol. . .	42.25	30.35	. .	47.20	56.50	. .	30.30	30.40	21.60	34.00	35.90	66.66	76.50	23.00	40.0	56.0	25.4	45-50	22-24
SiO_2 . . .	36.80	25.50	. .	39.30	49.70	. .	26.50	. .	18.15	. .	29.50	. .	53.45	. .	. .	. .	. .	. .	. .
H_2O . . .	. .	. .	. .	. .	. .	. .	. .	1.60	4.7	5.80	. .	29.80	. .	. .	. .	. .	. .	. .	. .
Ag, oz. p. t.	5.50	8.70	. .	3.00	3.90	. .	5.00	6.10	. .	4.70	4.00	3.40	2.80	14.00	7.6	3.0	5.6	. .	. .
Au, oz. p. t.	0.018	0.014	. .	0.016	0.010	. .	0.02	. .	. .	. .	. .	. .	. .	. .	0.032	none	0.0025	. .	. .

Note. — Analyses under A to F, inclusive, represent 1902; G, H, I, and J are Mr. Goodale's figures for 1896.

TABLE I. — ANALYSES OF BUTTE ORES. — *Continued.*

PLANT.	E.			F.		G.		F.	I.			J.	
	First-class ores.	Coarse concentrates.	Fine concentrates.	Silver ores.	Silver ores.	Smelting.	Concentrating.	Smelting.	Smelting I.	Smelting II.	Concentrating.	Smelting.	Concentrating.
Cu	8.0	6.0	9.0	0.30	0.40	20.26	6.50	9.80	29.95	26.25	8.10	0.55	0.10
Fe	13.2	18.6	1.42	5.90	7.90	14.42	9.55	26.00	18.65	16.40	10.10	5.10	4.90
Zn	. .	. .	. .	11.90	22.50	5.30	8.66	. .	7.25	2.34	. .	16.29	4.60
S	18.0	23.0	34.0	9.70	21.00	26.34	15.87	33.60	36.15	29.18	14.88	13.19	6.43
Mn	. .	. .	. .	7.43	1.70	0.45	0.09	. .	0.26	. .	. .	6.73	5.56
As	. .	. .	. .	0.20	0.10	1.66	0.94	. .	0.44	4.00	. .	trace	. .
Sb	. .	. .	. .			trace	0.20	. .	. .	. .	. .	0.05	. .
Pb	. .	. .	. .	3.50	8.20	trace	0.25	. .	. .	. .	. .	8.55	2.44
Al_2O_3	. .	. .	. .	7.70	2.90	2.85	. .	. .	. .	. .	. .	. .	. .
CaO	. .	. .	. .	4.00	2.10	. .	. .	. .	. .	. .	. .	. .	. .
MgO	. .	. .	. .	1.80	0.80	. .	. .	. .	. .	. .	. .	. .	. .
Insol.	54.0	45.0	28.0	51.00	30.40	28.72	54.80	28.70	7.26	21.50	63.00	43.22	66.50
SiO_2	. .	. .	. .	41.70	26.20	. .	. .	. .	. .	. .	. .	. .	. .
H_2O	. .	. .	. .	. .	. .	. .	. .	. .	. .	. .	. .	. .	. .
Ag, oz. p. t.	6.5	. .	. .	67.50	24.20	30.00	10.00	11.2	24.00	20.00	6.5	43.00	20.00
Au, oz. p. t.	n. d.	. .	. .	0.13	0.60	$0.44	$0.44	n. d.	n. d.	n. d.	n. d.	$0.80	$0.40

NOTE. — Analyses under A to F, inclusive, represent 1902; G, H, I, and J are Mr. Goodale's figures for 1896.

IV. METALLURGICAL TREATMENT — THE ROASTING OF ORES.

The metallurgical processes in the Butte district show much uniformity with regard to the general principles on which they are based. They comprise roasting, smelting, converting, electrolytic refining, and casting. There is considerable diversity in the details of practice.

First-class ore, forming about 10 per cent. of the product of the mines, is, as a rule, smelted raw, and furnishes the desirable coarse ore of the blast-furnace charge. Concentrates, coarse limestone, and converter slag are the basic fluxes added to slag the siliceous gangue. In one instance some first-class ore is roasted in lump-form in stalls. This makes it less refractory than raw ore, and, furnishing available iron, permits the running of slags richer in iron, and hence more fusible than would be otherwise possible.

Heap roasting, which was practiced in the early days, has been entirely abandoned.

In crushing, the copper-silver ores readily break into small grains, making often more fines than can be carried by a stall or even by a blast furnace. This necessitates separating part of the fines by screening, the screenings going into the fine-ore roasting furnaces.

The form and general arrangement of the *open stalls* now in operation are the same as given by Peters.[1] The management, however, differs slightly from the description given by him. In making the bed of decayed wood, first used by A. J. Schumacher at the Parrot works, no attention is paid to an open cross channel to start the fire.[2] While Peters prefers to close the front of the stall with large pieces of lump ore, at Butte it is closed down to 2 feet from the bottom with a plate of sheet iron, which is braced by iron cross bars to prevent bulging caused by the swelling of the ores while roasting. The plate is supported by blocks of wood, between which are placed small sticks and kindling. During the roast, when the contents of the stall have risen to the highest point and begin again to shrink on account of the slight clinkering which always takes place, the front plate gradually glides down to the floor. A stall holding 30 tons of first-class ore, with about 10 per cent. of copper and from 30 to 35 per cent. of sulphur, burns thirty days, the sulphur being reduced to 3.5 or 4 per cent.

Crushing and sampling the ore, running it to the stalls, and filling and emptying the stalls is done by contract.

The fine-ore furnaces are represented by four types, the Allen-O'Hara, the Wethey, the Pearce, and the MacDougall; long-hearth reverberatory furnaces used in early times having been given up entirely.

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[1] *Modern Copper Smelting*, 1895, p. 148.

[2] See Peters, *op. cit.*, Fig. 21, p. 151.

The Allen-O'Hara Furnace.[1] — This is one of the permanent improvements on the original O'Hara furnace.[2] The two superimposed hearths are each 9 by 90 feet; the six carriages with their plows make the circuit in three and three-fourths minutes. Thus the ore lying about 3 inches deep is stirred every thirty-seven seconds, and remains about six hours in the furnace. A test showed that 3.64 horse power was required to run the furnace. The ore charged is a mixture of concentrates and fines from first-class ore, usually in the proportion of $4\frac{1}{2}$: 1, containing about 35 per cent. of sulphur.

From 47 to 50 tons of charge is roasted in twenty-four hours, or 60 pounds per square foot of hearth area, the sulphur being reduced to 8 per cent. One hundred pounds moist ore gives 72.7 pounds of calcines and one pound of flue dust. A furnace is attended in an eight-hour shift by one-half carman, who brings the ore; one-half fireman; one wheeler, who removes the calcines to the smelting department, and one-eighth repair man during the day shift only. A partial analysis of an average sample of roasted ore gave: Cu, 10.8; SiO_2, 31.6; Fe, 38.0; S, 7.6; Ag, 7.3 ounces per ton. One of flue dust: Cu, 10.8; SiO_2, 34.4; Fe, 36.5; S, 8.2; Ag, 7.2 ounces per ton.

The Wethey Furnace.[3] — The furnaces in operation each have four superimposed hearths. The furnaces erected in 1894 had two sets of hearths, each 50 by 5 feet, with the rabble carriages running between them, the arms projecting from the inner sides into the hearth. The furnaces erected since 1894 have been increased to 65 by 10 feet, with the rabble arms protruding on both sides and carried on carriages on the outside of the furnace hearths. The tripping doors, which close the continuous hearth slot, are not being repaired as they wear out, the furnace seeming to do satisfactory work without them. The position of the fire box, which was built at first to heat the second floor, has been changed so that the flame may enter the top hearth and dry and kindle the moist concentrates and slimes as they enter the furnace. While the smaller size furnace, with a hearth area of 2,000 square feet, puts through at the present time 60 tons of raw material with 35 per cent. sulphur, or 60 pounds per square foot of hearth area, reducing the sulphur to 8 per cent. and consuming 5.5 per cent. of slack coal, the

[1] Peters, *Modern Copper Smelting*, 1895, pp. 200–205. Hofman, *Metallurgy of Lead*, 1899, pp. 186–190.

[2] Küstel, *Roasting Gold and Silver Ores*, San Francisco, 1880, p. 84. Fraser and Chalmers, *Catalogue*, No. 4, 1891, p. 135.

[3] Hofman, *Metallurgy of Lead*, 1899, pp. 195–198.

larger size furnace, with an area of 2,600 square feet, roasts in twenty-four hours 90 tons of ore, with 35 per cent. of sulphur, or 70 pounds per square foot of hearth area, reducing the sulphur to from 5 to 6 per cent. and consuming somewhat under 4 per cent. of slack coal. One fireman attends each furnace.

The improvement in the work of the longer furnaces is due to the heat from the fire box having to travel a longer distance before it escapes to the flue, and to the ore traveling in a direction opposite to that of the flame, whereby the heat is more fully utilized than with the shorter furnace.

The Pearce Turret Furnaces.[1] — Three kinds of turret furnaces are roasting ores, the single- and double-deck furnaces of R. Pearce, and the six-deck furnace of R. F. Pearce. A single-deck furnace 36 feet in diameter, with a 6-foot hearth (or a hearth area of 505 square feet) and two stirrer arms, roasts in twenty-four hours 14 tons of concentrates (Cu, 7 per cent.; Fe, 24 per cent.; Zn, 12 per cent.; S, 32 per cent.; SiO_2, 23 per cent.; Ag, 14 ounces per ton), reducing the sulphur to from 7 to 8 per cent. and burning 400 pounds of coal per ton of ore.

The percentage of flue dust is very low.

The 36-foot furnace does better work than the first one built, which has the same width of hearth (viz., 6 feet), but is only 30 feet in diameter. If, nevertheless, the tonnage appears to be comparatively low and the percentage of sulphur left in the roasted ore somewhat high, this must be attributed to the large amount of zinc present.

In the double-deck furnaces, in which the ore fed on the upper hearth drops through a slot in the roof onto the lower hearth, the floor of the upper hearth and the roof of the lower hearth are separated, it will be remembered, by an air space 2 feet high. A furnace with a 6-foot hearth (area 1,010 square feet) treats in twenty-four hours 30 tons of concentrates (59 pounds per square foot of hearth area), containing Cu, 12 per cent.; Zn, 8 per cent.; S, 35 per cent.; Fe, 25 per cent.; As, 3 per cent.; SiO_2, 16 per cent.; Ag, 20 ounces per ton, with a reduction of the sulphur to from 6 to 7 per cent., 400 pounds of coal being burned per ton of ore.

A furnace with a 7-foot hearth (area 121,829 square feet) treats 42 tons (69 pounds per square foot hearth area) with a similar elimina-

[1] Peters, *Modern Copper Smelting*, 1895, pp. 205–214; Hofman, *Metallurgy of Lead*, 1899, pp. 175–182.

tion of the sulphur and a fuel consumption of 182 pounds per ton of ore treated.

Comparing the pounds of ore roasted per square foot of hearth area in the two furnaces (59 and 69 pounds), an improvement of 15 per cent. in favor of the 7-foot hearth is seen — the degree of desulphurization is about the same in the two furnaces; the consumption of fuel, however, has been reduced from 400 to 182 pounds, or 55 per cent. The labor required to fire a single-deck furnace is the same as with the double-deck, and that for supplying ore and fuel and removing roasted ore and ashes from one double-deck furnace is, of course, smaller than from two furnaces with single hearths. Thus the double-deck furnace is a decided improvement over the single hearth as long, be it understood, as ores are treated which do not frit or form accretions on the hearth.

The six-deck furnace[1] resembles the double-deck furnace in the stirring mechanism, and with it in the passage of ore and gases; it has one fireplace (3 feet by 3 feet 2 inches) on the top hearth, and one 4 feet 3 inches by 3 feet 2 inches on the bottom hearth.

It differs from the double-deck furnace in the number of hearths, 6 *versus* 2; in the fact that there are no air spaces between the superimposed hearths, the roof of one forming the floor of the next one above; and that the portions of the inner wall above the horizontal slots of hearths one to five have cantilever supports with trussed cords projecting from the central column. This makes rather a complicated structure, requiring a large number of iron parts of various sizes, some of considerable weight. The iron parts of the furnace weigh 165 tons. The furnace is 32 feet in diameter and 20 feet 9 inches high; the hearths, 7 feet wide, are 2 feet $9\frac{1}{2}$ inches from roof to roof, which is $4\frac{1}{2}$ inches thick; the sand filling forming the working hearth is 5 inches thick at the crown of the roof and 12 inches at the sides; the distance from the working hearth to the inner slot (protected by 1-inch castings) is 6 inches, and to the springings of the arch 16 inches; the rise of the arch is 8 inches. A hearth has four stirring arms making one revolution in sixty-five seconds. The horizontal flues in the outer walls are omitted, through which air passes in the older furnaces in order to be warmed and admitted to the surface of the roasting ore. Around the furnaces are platforms to facilitate access to the hearths. The moist concentrates, dumped into the shoe of a bucket elevator, are

[1] *United States Patent*, No. 671,071, April 2, 1901.

raised to the top of the furnaces and discharged into a hopper, partly closed at the bottom by a corrugated roller feed; this delivers into a slot 8 inches wide by 7 feet long in the roof of the top hearth. The furnace roasts 56 tons ore in twenty hours, or 38 pounds per square foot of hearth area; it makes 4 per cent. of flue dust on the weight of the ore charged, and requires 28.5 pounds slack coal per ton of ore. The quality of the work is similar to that of the double-deck furnace; the amount of ore roasted per square foot of hearth area is a great deal smaller, but the saving of fuel is much larger. The low fuel consumption brings out in a striking way the benefits that are derived by diminishing the loss of heat by radiation.

The MacDougall Furnaces. — This is a vertical cylindrical furnace with arched horizontal hearths having discharge openings alternately near the centre and periphery, and a central revolving shaft with horizontal radial stirring arms, which are provided with teeth placed at an angle to the arm. The teeth of the arms on the even-numbered hearths are set in a direction opposite to that of the odd-numbered, thus on the odd-numbered hearths they push the ore toward the centre, and on the even-numbered toward the side. The ore, fed mechanically onto the furnace, is delivered continuously onto the outer edge of the first hearth, where the teeth of the stirring arm turn it over and work it toward the centre, where it drops through an annular opening onto the second hearth; the arms of this turning the ore over and over will convey it slowly to the openings near the periphery, through which it drops onto the third hearth and so on until it arrives on the closed bottom, where it falls through drop holes into a hopper.

The ore spread evenly over the hearth thus travels in zigzag through the furnace, passing from periphery to centre and *vice versâ*. It is warmed at the top of the furnace; it then ignites and burns freely; and lastly most of the sulphates formed are decomposed lower down. The air necessary for oxidation is admitted through doors in the side, most of it entering on the bottom hearth. The gases, and with them the dust, pass upward through the ore-discharge openings, and are drawn off through pipes in the roof. In starting, the furnace has to be heated up to the kindling temperature of the ore, which, if rich enough in sulphur, will burn of its own accord without the use of any carbonaceous fuel. If too low in sulphur, the heat necessary for successful roasting has to be supplied by one or more external fireplaces.

Three kinds of MacDougall furnaces interest us here: the Original MacDougall, the Herreshoff, and the Evans-Klepetko.

The Original MacDougall Furnace.[1] — This furnace, patented by MacDougall Brothers in the beginning of the seventies, was in operation in Liverpool, England, burning pyrites to furnish sulphurous gases for the manufacture of sulphuric acid. It had to be abandoned on account of the large amount of flue dust formed and the many mechanical difficulties encountered. The furnace, 6 feet in diameter and 12 feet high, was built of seven cast-iron cylinders bolted together, and had six closed chambers. The vertical solid shaft, 6 inches in diameter, had cast-iron stirring arms. A 2-horse power engine was sufficient to drive the mechanism. The furnace roasted in twenty-four hours $3\frac{1}{2}$ tons pyrites, or 0.020 tons per square foot of hearth area, reducing the sulphur to 2 per cent.

The Herreshoff Furnace. — Most of the Herreshoff furnaces in operation at Butte have a $\frac{1}{4}$-inch steel casing; they are 10 feet 10 inches in diameter and 11 feet 6 inches in height, and are lined with a full course of red brick. A few furnaces built of two full courses of brick and hooped with iron bands only are doing better work in cold weather than the furnaces that are sheeted and lined with a single course, thus showing that with ores running low in sulphur, when heat has to be husbanded, an 18-inch side wall has its advantages. A furnace has five arched brick hearths $4\frac{1}{2}$ thick and 12 and 13 inches apart; the pitch of the arch is $4\frac{1}{2}$ inches; a current of air passes through the hollow central revolving shaft, which is 14 inches in diameter. The hollow horizontal cast-iron stirring arms, two to a hearth, have seven and eight teeth, respectively; they are easily exchanged when worn out.

The top hearth acts as a dryer; the second starts the roasting; on the third the ore roasts freely (innumerable sparks are seen); on the fourth there are no sparks; and on the fifth the ore looks dark. A furnace, with its shaft making fifty revolutions per hour, roasts in twenty-four hours from 5 to 6 tons of wet concentrates with about 35 per cent. of sulphur, or 0.015 tons per square feet of hearth area, reducing the sulphur to 6 per cent. Seventy furnaces make from 4 to 5 tons, or 6.4 per cent. of flue dust.

In Eastern sulphuric acid plants this furnace[2] treats in twenty-four

[1] Lunge, *Sulphuric Acid and Alkali*, London, 1891, I, pp. 260–264. Also, *The Mineral Industry*, 1898, VI, p. 235.

[2] *The Mineral Industry*, VI, pp. 235–238.

hours from 3 to 4 tons of pyrites, with 44 per cent. of sulphur, or 0.0098 tons per square foot of hearth area, reducing the sulphur to from $2\frac{1}{2}$ to $3\frac{1}{2}$ per cent.; the shaft makes thirty revolutions per hour.

The Evans-Klepetko Furnace.[1] — Shortly after the Herreshoff furnace had proved a success, a larger furnace,[2] 16 feet in diameter, was constructed on the same general lines; it had eight hearths, an air-cooled shaft, and eight solid cast-iron arms. The heat generated was sufficient to cause the arms to bend under their own weight. Air-cooling having proved unsatisfactory, water-cooling was substituted, and the number of hearths reduced from eight to six. The leading differences then between the Evans-Klepetko and the Herreshoff furnaces are the size and the water cooling of the central shaft and the stirring arms. The furnace is 18 feet $3\frac{1}{2}$ inches high and 15 feet 10 inches in diameter; it is sheathed with $\frac{3}{8}$-inch boiler iron, and lined with a full course of red brick; it has six arched hearths with a 9-inch spring and 3 feet apart; each hearth has two stirring arms making sixty circuits per hour. The furnace stands on columns 12 feet high to allow the roasted ore to be collected in hoppers and discharged into cars. Six (resp. eight) furnaces form a battery; they are placed from centre to centre 18 feet apart in one direction and 21 feet 3 inches in the other. Each furnace has two gas flues, 2 feet in diameter and 12 feet apart, passing out of the roof; the flues from three furnaces lead into one main 6 feet (resp. 7 feet) in diameter, having openings along the top and bottom for removing the flue dust; the bottom openings have vertical pipes which are connected directly with the calcine hoppers. The central shaft of a furnace is driven from below. Three (resp. four) furnaces receive their motive power from one main shaft, with which they are connected by friction clutches. A clutch can be thrown in or out from each of the three platforms surrounding a furnace. The cooling water is forced down to near the bottom of the revolving hollow shaft (9 inches inner diameter) through a 3-inch pipe and out to the ends of the horizontal stirring arms through 1-inch horizontal pipes. In its upward passage between shaft and pipe it takes up the return water from the stirring arms, and discharges at the top through two spouts into a stationary launder. Shaft and arms are made

[1] *United States Patent*, No. 700,399, May 13, 1902; Drawing, see *Canadian Mining Review*, April 30, 1903.

[2] C. H. Repath, "The Mechanical Engineer, a Factor in Modern Mining, Milling, and Smelting," Mining Congress, Butte, Sept. 3, 1902, through *Mining and Scientific Press*, Sept. 13, 1902.

up of flanged sections to permit easy exchange. Running the overflow-water at 80° C., 20 gallons of cooling water per minute are required by a furnace. Forty-eight furnaces are supplied with the necessary draft by a steel stack 200 feet high, lined throughout (inner diameter 20 feet; outer, 23 feet 6 inches). A 30-horse power motor is provided to run the machinery. Tests have since shown that 10 horse power is required for six furnaces. When ore too low in sulphur to be self-burning, *i. e.*, about 28 per cent., is to be roasted, one auxiliary fireplace for two furnaces, placed on the level of the bottom floor and delivering the flame onto the fourth floor, furnishes the additional heat required.

The moist concentrates, arriving from the ore-dressing plant on an overhead track, are dumped into the two feed hoppers of a furnace. These are 10 feet apart, and are 4 feet 6 inches in diameter for 9 feet 6 inches, and then conical for 7 feet 6 inches. At the bottom they are made 3 feet 6 inches by 1 foot 2 inches to fit into the openings in the roofs of the furnaces. The top of the hopper has a grating to break up any lumps and to keep out large pieces that may have accidentally gotten into the ore. The hopper holds two cars of concentrates, or 33 tons. This weight, pressing downward, prevents any hanging of the charge, and thus solves the difficulty of feeding moist fine concentrates. The contents of the hopper are continuously discharged into the furnace by means of a feeding device of the form of an I, which passes to and fro under the mouth with a throw of about 10 inches. The rods of the two feeders are joined by a cross-piece, which is pivoted in the centre and connected by a rod to one end of a link, while the other end is attached to the pitman of a crank and spur wheel; the spur wheel is geared to the central shaft of the furnace. The ore is spread on the top hearth to the thickness of 3 inches by the stirring arms. The tops of these arms are protected by cast-iron caps against rapid wear at the places where they strike the ore column coming down through the feed openings. The outer half of a hearth is 9 inches thick, the inner 6 inches. The reasons for the difference are: saving of brick, reducing of weight at centre, preventing contact with stirring teeth when the hearth rises upon heating, and furnishing an offset to hold the loose working bottom (crushed limestone, sometimes flue dust and even coke dust) which is about 4 inches thick at the centre and 9 inches at the periphery. The two arms of a hearth have seven and eight cast-iron teeth; these are 8 inches long by 6 inches wide by $\frac{5}{8}$ inch thick; the lower 3 inches of the teeth, which come in contact

with the ore, are chilled. There is no advantage in making the teeth thicker than $\frac{5}{8}$ inch, as they wear off obliquely to the working surface, and consequently would simply offer a larger wearing surface but not lengthen the life. At the same rate that the teeth are worn off the ore builds on the loose working hearth. When they are to be renewed they are removed, and a plow is slipped over the stirrer arm and moved a little toward the centre after every circuit of the arm. Thus a crust is easily and surely broken up. The crusts on the second hearth are harder to break than those of the other hearths. The teeth on the top hearth last from twenty-five to thirty-four days; those on the sixth, from six to eight months. The shorter life of the former is attributed to mechanical wear caused by decrepitation of the ore. It cannot be due to chemical action, as the top hearth serves mainly for drying, unless the moisture should cause some sulphuric acid to form; but the temperature of the escaping gases is 315° C., and the draught is sufficiently strong to show a depression of 0.3 inches of water.

Roasting begins on the second or third hearth, varying with the strength of the draught and the number of circuits that the stirring arms make. With a strong draught the heat creeps up; with one circuit in fifty-five seconds the roast begins on the second hearth; with one in seventy-five seconds on the third. As the ore passes from the first hearth through the annular opening at the centre of the roof of the second hearth (protected by a cast-iron ring), it strikes a distributor which prevents any of it from passing down the shaft. During normal work three doors on the bottom hearth are left open for the admission of air. If the furnace becomes too hot, the doors on the third floor are opened more or less, and thus the draught checked. While the ore drops from hearth to hearth, the upward gas current carries with it dust, and this striking the roof adheres to it in part and builds. In order to protect the arms from being worn out at these places, they are protected by cast-iron caps. When the dust has grown to a thickness of 4 inches it is removed with chisel-pointed bars. Experiments with cast-iron plates to protect the roof at weak spots have proved successful, as the dust is easily pried off. The chunks of fritted dust go to the blast furnace as welcome lump ore; they contain only 6 per cent. of silica with ore of 30 per cent. silica, thus showing that light particles of ferric oxide are more easily carried away by the gas currents than heavy gangue. The dust collected in the settling chambers runs higher in sulphur than the roasted ore.

In starting a furnace crushed limestone is fed to form the working

bottom. Then a small fire of dry, soft, *i. e.*, long-flame, wood is started from the three side doors on the third and fifth hearths. A new furnace is brought to a dark red in from three to four days; an old furnace requires only two days. Now concentrates are fed. After charging for from five to six hours, it sometimes happens that the furnace cools down too much, and this makes it necessary to start on the third and fifth floors a new fire for from one and one-half to two hours; occasionally feeding of the ore is stopped and half a ton of coal is charged. When this has come to the third hearth the machinery is stopped and the coal is allowed to burn out. The kindling of the coal may have to be assisted by placing dry wood on the hearth. Under normal conditions a furnace does its best work when the flue shows a depression in water of 0.3 inch; if it is less, the furnace gets cool. The normal ore mixture passes through the furnace in about one and one-half hours. If the furnace gets too hot, more slimes are fed; if too cold, coarser concentrates are charged, and the rate of feeding is reduced. As indicated above, the temperature is further regulated by the admission of air; closing the bottom doors drives up the heat, opening them draws it down; opening doors higher up checks the draught. The rate of feed when once settled upon is usually not altered, and the number of circuits the arms make per hour remains, of course, constant.

The dust, which collects in the flue connecting three (resp. four) furnaces and forms from 4 to 5 per cent. on the ore, is raked out every day; sometimes part of it is blown into the main flue of a battery with a hose under a pressure of about 90 pounds per square inch. The loss of weight in ore, including flue dust, is about 20 per cent.

A furnace treats under normal conditions in twenty-four hours 40 tons sulphide ore, with 35 per cent. sulphur and 10 per cent. copper, or 0.042 ton per square foot of hearth area, reducing the sulphur to 7 per cent.; roasted ore, with 14 per cent. copper treated in the same manner, retains about 10 per cent. of sulphur. The product can, of course, be varied with the speed of travel of the stirring arms. Thus with one circuit in seventy-five seconds it is reduced to 30 tons; with one in fifty seconds it is increased to 50 to 55 tons.

A section of 6 (resp. 8) furnaces is tended to in an eight-hour shift by one-third foreman, one furnace man, one helper, one-sixth oiler, one-ninth repair man, and one trimmer.

The following partial average analysis of roasted ore represents two determinations from the average day-and-night samples, taken during an experimental run of fifteen days:

SiO_2, 26.9 per cent.

Cu, 18.3 per cent., of which 9.9 was present as CuO.

Fe, 30.0 per cent., of which 17.9 was present as FeO.

S, 9.2 per cent., of which 0.81 was present as SO_3.

The matte formed in crucible fusions contained 65 per cent. copper. In comparing the roasted ore of the MacDougall furnace with that of the Brückner furnace[1] of the same plant, it is found that it is reddish brown as against a dark brown (more Fe_2O_3 is formed), more porous (1 cubit foot weighs 87 pounds as against 100 pounds) and finer (the rolling of the ore in the Brückner cylinder causes particles to form pellets). In regard to flue dust, the MacDougall furnace makes much less than the Brückner cylinder, although the amount is still great; they compare as 1 : 3 to 4.

General Results.—Some of the leading facts of the roasting furnaces operated in Montana are brought together in the subjoined table:

TABLE II.—DETAILS OF MONTANA ROASTING FURNACES.

	Tons Roasted 24 Hours.	Horse Power Required.	Sulphur in Concentrates.	Sulphur in Calcines.	Area of Hearths.	Concentrates per Sq. Ft. Hearth.	Coal per Ton Concentrates.	Cost per Ton.
			Per cent.	Per cent.	Sq. ft.	Lbs.	Lbs.	
Hand reverberatory furnace, 69½ ft. x 16 ft. hearth	13	. .	35	7-8	1,112	12	307	$2.00
Allen-O'Hara furnace, two hearths, 94 ft. x 9 ft.	51	3.64	35	8	1,692	77	145	0.78
Brückner cylinder, 8 ft. x 16 ft.	18-20	1½	37	9.5	. .	. .	540*	1.25
Pearce single-deck furnace	14‡	1½	32	7-8	505	55	400	0.98†
Pearce double-deck furnace, 6-ft. hearths	30‡	3	35	6-7	1,010	59	400	0.98†
Pearce double-deck furnace, 7-ft. hearths	42‡	3	35	6-7	1,218	69	182	0.98†
Keller-Gaylord-Cole furnace, two sets of six hearths	50	1¾	38	7-10	2,592	38	67	. .
Wethey furnace, two sets of four hearths, 50 ft. x 5 ft.	60	4	40	8	2,000	60	110	. .
Wethey furnace, two sets of four hearths, 65 ft. x 10 ft.	90	4	35	5-6	2,600	70	80	. .
Herreshoff furnace, five hearths	5-6	. .	35	6	135	80	. .	0.50
MacDougall-Evans-Klepetko furnace, six hearths	40	1¾	35	7	952	84	. .	.35
Pearce multiple-hearth furnace, six hearths	56‡	12	35	6-7	2,947	38	28.5	0.98†

* Data obtained from operations of six months at Great Falls. For quality of coal see analysis of Sand Coulee and Belt coal elsewhere in this paper.

† Average.

‡ These low figures are due to the character of the ore (Gagnon mine) which carries 8-12 per cent. zinc.

[1] Size: 8 by 16 feet, one revolution in seven minutes; 15-ton charge in twenty-four hours; drying, two hours; burning, four hours; charging, roasting, and discharging, eighteen hours.

V. Smelting of Ores in the Blast Furnace.

The leading data of the blast furnace practice are brought together in Table III. The furnaces are all of the same general type; they are oblong and water-jacketed, and have an internal crucible which discharges the slag-matte mixture continuously over a raised spout, whereby the blast is trapped. The matte is settled in a large fore-hearth, while the slag overflows, to be granulated and removed by water or to be collected in waste-slag cars and conveyed electrically to the dump and poured.

The general tendency is to increase the size of the furnaces. The smallest distance between tuyères is 42 inches, the largest 56 inches; in one instance it was increased to 72 inches. This worked well with coarse ore, but it proved a failure with the usual run of ores which contain many fines. Thus a screen analysis of an average sample of blast furnace ore from one important mine gave:

300 pounds ore over 2 inches	13.3 per cent.
450 pounds ore 2–1 inches	20.0 per cent.
500 pounds ore 1–$\frac{3}{8}$ inches	22.2 per cent.
1,000 pounds ore under $\frac{3}{8}$ inch	44.5 per cent.
2,250 pounds	100.0 per cent.

showing why such a large distance between tuyères proved impracticable. The greatest length of furnace at tuyères is 180 inches; this seems to be the limit with the present mode of construction of jackets. The height of the furnaces increases with the distances between tuyères from 7 to 18 feet. But the charge in the 18-foot furnace is kept from 4 feet to 8 feet below the throat, so that the actual working height is reduced to from 10 to 14 feet. All furnaces are run with a top more or less hot. Temperature measurements of furnace gases in one instance showed 315° C., and a draught measurement gave a depression 0.8 inch of water. The water jackets are all made of low-carbon steel. They are two-high, reaching to a short distance below the feed door, the remaining space being of brick. There are usually two jackets on a side and one on the ends; the lower tiers of side jackets only have a bosh, the amount of which is very small, while the upper tiers are vertical, as are generally the end jackets. The gases are always drawn off above the feed floor. In one instance the top of the furnace carries a Giroux blast heater, made by the Union Iron Works of San Francisco, which raises the temperature of the blast to 150° C., and at the same time

TABLE III.—Examples of Montana Blast Furnace Smelting.

Furnace.	A.	B.	C.	D.	E.	F.
Horizontal section at throat	66 in. x 112 in.	54 in. x 96 in.	72 in. x 180 in.	53 in. x 160 in.	72 in. x 180 in.	72 in. x 180 in.
Horizontal section at tuyères	42 in. x 112 in.	44 in. x 96 in.	56 in. x 180 in.	42 in. x 160 in.	56 in. x 180 in.	56 in. x 180 in.
Area at tuyères	32.66 ft.	29.329 ft.	70.029 ft.	46.629 ft.	70.29 ft.	70.29 ft.
Height, tuyères to throat	7 ft. 3¾ in.	9 ft.	18 ft.	10 ft. 6 in.	18 ft.	18 ft.
Height, tuyères to top of crucible	16 in.	9 in.	11 in.	9 in	11 in.	11 in.
Water jackets, height	9 ft.	6 ft.	14 ft. 10 in.	10 ft. 6 in.	14 ft. 10 in.	14 ft. 10 in.
Bosh, inches in feet	12 in. in 6 ft. 4 in.	7 in. in 5 ft.	8 in. in 7 ft.	1 in. in 10 in.§	8 in. in 7 ft.	8 in. in 7 ft.
Crucible, depth	8 in.	16 in.	27 in.	9 to 14 in.	28 in.	28 in.
Forehearth, fixed or movable	Fixed.	Fixed.	Fixed.	Fixed.	Fixed.	Fixed.
Forehearth, shell dimensions	8 ft. x 5½ ft. x 3 ft.	12 ft. di. 52 in. deep.	14 ft. di. 56 in. deep.	7 ft. 3 in. x 10 ft. 6 in. x 4 ft.	14 ft. di. 56 in. deep.	14 ft. di. 56 in. deep.
Tuyères, number	16	12	32	18	32	32
Tuyères, diameter	3 and 3½ in.	2½ in.	3 in.	4 in.	3½ in.	3½ in.
Tuyère ratio*	3.5	2.01	3.23	4.85	3.79	3.79
Blast, pressure, oz.	16†	24	35	30 to 32	28	28
Blast, cu. ft. per ton of ore	57,000	68,574	65,000	. .	. .	60,176 ‖
Blast, temperature, degrees C.	150	15	15	15	15	15
Charge (ore + flux), weight, lbs.	3,300	2,400	2,150	4,000	8,700	10,000
Charge, tons in twenty-four hrs.	125	165	400	220	400	450
Charge, tons per sq. ft. hearth area	3.82	5.56	5.71	4.71	5.71	6.43
Coke, per cent. ash	20.0	21.3	9 to 15	20.0	8.0	8.0
Coke, per cent. of charge	13	12.3	6.1	11.5	9.2	8.5
Men, number in 8-hr. shift‡	5 + 2	4 + 2	3 x 2½	4½ + 3½	3 +	4 +
Matte, per cent. of charge	. .	5.1	. .	. .	. .	5.5
Matte, Cu per cent.	50	46	49	38	42	42.12
Zn per cent.	. .	. .	1.0	. .	. .	. .
Ag oz.	69	52	20	. .	. .	. .
Au $	2	10	. .	. .	. .	. .
Sp. gr.	. .	. .	5.0 to 5.1	. .	. .	4.8
Slag, SiO_2	47.7	44.3	40.7	48.0	40.0	42.6
Fe(Mn)O	18.5	27.0	26.2	17.0	23.0	30.7
Ca(Mg)O	28.2	23.0	19.4	20.0	22.6	19.5
Al_2O_3	. .	9.0	9.6	6.0	9.8	. .
Cu	0.35	0.32	0.25	0.30	0.30	0.25
Ag	0.43	0.30	. .	. .	0.20	. .
Sp. gr.	. .	. .	3.0 to 3.55	. .	. .	3.3

* Sq. in. tuyère area : 1 sq. ft. hearth area.

† Estimated at 50 per cent. of the rated capacity of the blowers.

‡ First figure represents the regular crew ; second figure auxiliary labor.

§ Lower jackets only.

‖ Ore-flux.

acts as a dust catcher, delivering the saved dust back into the blast furnace. The charge in most works is introduced by hand from the sides of the furnace; occasionally it is fed from the ends. In some instances, with large furnaces putting through 400 tons charge in twenty-four hours, mechanical feeding has taken the place of hand feeding. The charge is brought by electric or compressed-air traction to the side of the furnace on trucks with tilting carriages; the doors are raised by compressed air, and the charges slid in alternately from one side and the other. This method seems to be satisfactory. At least, worked out in one large plant, it has been introduced in another with some improvements in the construction of the carriage. While it is generally maintained, and has recently been again emphasized by P. Johnson[1] in connection with copper smelting, that the successful working of a blast furnace is much governed by the care given to feeding, it must be admitted that, considering the mechanical feeding practiced at Butte with copper ores producing siliceous slags high in lime, and at Pueblo, Colorado,[2] Murray, Utah,[3] East Helena, Montana,[4] with lead ores, where blow holes or uneven sinking of charges must be absolutely avoided, hand feeding cannot hold its former monopoly, but will have to yield to mechanical devices.

The hearth in some plants is supported by jackscrews; in others it is built up solid from the foundations. It rises some distance above the furnace floor in order to furnish the height necessary for the forehearth or settler and the disposal of the waste slag. The hearth is lined to a depth of from 24 to 30 inches with fire brick, or only with one or two courses of fire brick, and then with a mixture of quartz and clay. The crucibles have a depth varying from 8 to 28 inches. Although deep crucibles last rarely longer than three months and shallow crucibles have a longer life, the former are more common. Preliminary experiments with lining deep crucibles with chrome brick[5] have given promising results in preventing breakouts of matte. There appears to be no *a priori* reason for having a continuously discharging crucible as

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[1] *Engineering and Mining Journal*, Aug. 23, 1902.

[2] Hofman, *Metallurgy of Lead*, 1901, p. 216; *Transactions American Institute of Mining Engineers*, November, 1901.

[3] *Engineering and Mining Journal*, June 28, 1902.

[4] *Transactions American Institute of Mining Engineers*, November, 1901.

[5] Lang, *Engineering and Mining Journal*, Jan. 23, 1897. Packard, *Engineering and Mining Journal*, Feb. 13, 1897. Glenn, *Transactions American Institute of Mining Engineers*, XXXI, p. 374.

deep as 28 inches; 8 to 10 inches would seem to be sufficient to furnish the matte bath necessary to cover the bottom satisfactorily, but this is disputed by the best of authority. In treating zinckose ores the deep crucible would probably soon fill up with mushy matte and cause infinite trouble, but the ores smelted fortunately contain very little zinc.

It is claimed that with a deep crucible a furnace puts through more charges, but this has so far not been proved. As far as the strength of the jackets is concerned, the deep crucible is preferable, as with a shallow crucible the tuyères are located higher up (viz., 24 and 12 inches; see table) in order to furnish a smelting zone of some height, and the water-cooled breast has to be built into the jacket; while with the deep crucible it is placed underneath the jacket.

The slag-matte mixture is usually discharged from the centre of one of the sides of a furnace. The water-cooled tymp is made of cast iron or, preferably, of copper. Experiments with copper and copper alloys gave the following data:

Life, Days.	Cu. Per Cent.	Zn. Per Cent.	Al. Per Cent.	P. Per Cent.	Sn. Per Cent.
365	100	. .	. .	Some.	. .
144	90	. .	0.5	. .	10
75	89	3.0	0.25	. .	8

The solid cast-iron spout with water-cooled nose is very common; wrought-iron pipe surrounded by cast iron has proved unsatisfactory; a water-cooled wrought-iron spout with a cast-copper removable nose lasts about 120 days, and is often preferred to cast iron.

The tuyère-stock in common use consists of a cast-iron tuyère-box firmly attached to the jacket, and connected by a sheet-iron tuyère-pipe having a gate with the bustle pipe.

The large size of the roofless forehearth in which the matte settles out from the overflowing slag is a development of Montana practice, caused by the necessity of storing considerable quantity of liquid matte for the converters. As far as the settling of matte is concerned, oval and oblong hearths are sufficient, as the freedom of copper in waste slag depends to a large extent upon the length of the path the slag can travel before it overflows. In regard to storage capacity and strength of vessel the circular hearth is preferable, hence it is only with a comparatively small blast furnace not furnishing direct matte that we find

the oval or oblong forehearth. The circular forehearth, 14 feet in diameter and 56 inches high, with a 9-inch brick lining and a 9-inch brasque backing, holds about 60 tons of matte when new. In order to prevent breaking out, the shell is sprayed with water. After being about three months in use, its capacity has so decreased by building that it becomes necessary to renew it. The 9-inch course of brick sometimes has a 3-inch backing of sand, which seems to be sufficient. The smaller oblong hearth has, as a rule, cast-iron water jackets on the sides; it has a 9-inch bottom, which crusts up about 6 inches; the sides are lined with a half-course of brick, a 1-inch air space being left at the water jacket. All forehearths have, of course, a tapping slot for matte. It is closed in a few instances by a cast-iron plate, but more commonly by a solid copper plate with tap hole. The matte is tapped from the forehearth into a ladle, which discharges its contents either into flat cast-iron moulds holding about 3,000 pounds, or into the converter. The practice of bringing the converter to the forehearth of the blast furnace has become obsolete. The blast-furnace forehearth, if of large size, is of great assistance to the reverberatory matting furnaces in a plant which converts matte, as it makes little difference whether the forehearth contains much or little matte; it can be allowed to fill to near the top, or it can be tapped nearly dry. With the reverberatory furnace this is not the case. The aim there is to allow the level of the matte to fluctuate as little up and down as possible. This means that only a certain quantity of matte can be tapped at stated intervals. If, therefore, the converter calls for matte and the reverberatory furnace can furnish only an insufficient amount, there is the blast-furnace forehearth, forming a kind of balance wheel, to help over the difficulty. The waste slag is caught in tilting waste-slag pots of from 5 to 6 tons capacity, and hauled to the dump, or it is granulated. Near the bottom of the launder carrying the slag granules there is found often a mechanical sampler, driven by the granulating water, which takes out the whole of the stream of granules at stated intervals, and thus furnishes a true sample of all waste slag made.

Coke. — The coke used around Butte comes from different parts of the country, as shown by the following table; most of it, however, is made from Montana coal:

TABLE IV.—Average Coke Analyses.

Year.	Total H_2O.	H_2O.	Volatile Matter.	Fixed Carbon.	Ash.	Sp. Gr.
Lehigh, Pennsylvania:						
1897	. .	0.3	1.3	85 7	12.6	. .
1898	. .	0.5	3.2	82.7	13.4	. .
1899	. .	0.4	3.9	81.0	14.6	. .
1900	0.6	0.2	4.2	81.8	13.8	. .
1901	0.3	0.2	3.6	84.2	12.2	. .
Total	0.9	1.6	16.2	415.4	66.6	. .
Average	0.4	0.3	3.2	83.1	13.3	. .
Horr, Montana:						
1899	. .	0.5	4.2	75.2	20.3	. .
1900	. .	0.3	4.0	75.7	19.9	. .
Total	. .	0.8	8.2	150.9	40.2	. .
Average	. .	0.4	4.1	75.4	20.1	. .
Crow's Nest, British Columbia:						
1900	0.5	0.2	4.1	85.8	9.9	. .
1901	0.3	0.2	3.3	85.3	10.7	. .
1902	. .	0.2	4.7	84.5	10.5	. .
Total	0.8	0.6	12.1	255.6	31.1	. .
Average	0.4	0.2	4.0	85.2	10.3	. .
Belt, Montana:						
1899	. .	2.3	4.7	79.3	13.7	. .
1902	. .	0.3	3.6	83.0	12.4	1 45
Total	. .	2.6	8.3	162.3	26.1	. .
Average	. .	1.2	4.2	81.1	13.0	. .
Cardiff, Colorado:						
1902	. .	0.4	3.6	85.7	10.3	2.45
West Superior, Wisconsin:						
1902	. .	0.3	3.7	81.4	14.5	. .
Fairmount, West Virginia:						
1902	0.1	0.1	4.5	82.4	13.0	. .
Castle Gate, Utah:						
1902	. .	0.2	3.5	86.1	10.2	. .

TABLE V.— ANALYSES OF COKE ASH.

SOURCE.	KIND.	SiO_2.	Fe_2O_3.	Al_2O_3.	CaO.	MgO.	S.
Montana	Horr	57.0	19.0	21.2	3.3	. .	. .
Montana	Belt	51.6	10.0	33.6	. .	. .	. .
Colorado	Cardiff	48.9	18 9	23.9	5.8	1.4	0.8

In the management of a blast furnace the tendency has been toward large tonnages, with the production of 50-per cent. copper matte, and slag low enough in copper to be a waste product. Tonnage had to contend with siliceous ores low in iron and with limestone as the only available flux, which caused the making of slags of high melting temperatures. As, further, large amounts of limestone are required to make slags that are at all fusible, the formation temperatures lie higher than when there is a more intimate contact of acid and base. Tonnage was therefore obtained mainly by increasing the size of the furnace, as shown in the preceding table. The combination of siliceous slags low in iron, with much air, has been the cause of as much as 70 per cent. of the sulphur in the charge being burnt off in its descent through the furnace. But even with the large elimination of sulphur in a comparatively dense charge, it is often not possible to obtain the 50-per cent. matte desired for the converter plant; hence blast-furnace matte is sometimes tapped from the forehearth into a ladle, and poured into the reverberatory matting furnace to raise the percentage of copper by mixing with higher-grade matte. Table VI gives in percentages some idea of how the charges are made up.

Table VII shows partial analyses of flue dust and briquettes, giving an idea of their general character.

The briquetting machines of Chisholm, Boyd and White (Chicago), and of H. S. Mould (Pittsburg) are both used.

VI. SMELTING OF ORES IN REVERBERATORY MATTING FURNACE.

The leading facts of the reverberatory furnace practice are brought together in Table VIII.

The characteristics of Montana reverberatory matting furnaces are the form of hearth and the greatness of its area. While as late as 1891 the hearth had the usual pear-shaped form of the original Welsh

TABLE VI.—Percentage of Components of Charge.

Plant.	First-class Ore.	Coarse Concentrates.	Matte.*	Converter Slag.†	Briquettes.‡	Limestone.	Coke.
A	22.4	20.5	4.0	17.8	4.0	22.8	8.5
B	52.5	. .	2.3	8.9	. .	25.6	10.7
C	15.5	25.0	22.5		0.2	27.6	9.2

*The matte comes in part from the reverberatory matting furnace, which had to be tapped when the converter plant was not ready. In part it is purchased matte to be remelted for the converter.

†The converter slag always goes to the blast furnaces, as it furnishes coarse material, aids the smelting in having been already once smelted, and acts as a basic flux (see Analyses below). Experience has shown that it not only does little good in the reverberatory matting furnaces, but that it does harm in that it increases the amount of slag to be skimmed, and reverberatory-furnace slags run higher in copper than blast-furnace slags.

‡The briquettes are a mixture of flue dust with 3 to 4 per cent. slacked lime; sometimes slimes from the ore dressing works are added to replace the lime.

TABLE VII.—Analyses of Flue Dust and Briquettes.

Flue dust from	Cu.			SiO_2.	Fe.	Al_2O_3.	CaO.	S.		As and Sb.	Ag, oz. per Ton.
	Total.	Sol. in H_2O.	Sol. in H_2SO_4.					Total.	Sol. in HCl.		
Flue near MacDougall furnace	13.3	. .	. .	47.9	11.9	. .	. .	12.0	. .	. .	4.2
MacDougall furnace	11.8	. .	. .	52.0	11.0	. .	. .	7.9	. .	. .	4.8
Blast furnace	10.6	. .	. .	33.2	18.7	. .	. .	10.7	. .	. .	5.3
Blast furnace	10.4	4.2	5.7	26.5	13.9	37.7	4.2	7.3	5.3	1.6	. .
Blast furnace	12.85	. .	. .	34.8	16.3	. .	. .	13.3	. .	. .	. .
Dust chamber	13.3	1.6	5.4	31.7	8.3	15.8	0.5	7.6	. .	9.1	. .
Dust chamber	7.2	. .	. .	34.3	11.9	13.3	. .	7.27	. .	. .	3.4
Stack	. .	. .	. .	. .	. .	. .	. .	. .	. .	14.3	. .
	6.3	. .	. .	40.0	20.8	. .	. .	9.3	. .	. .	54.0
Briquettes (dust and slimes)	6.5	. .	. .	63.0	13.0	5.3	. .	8.0	. .	. .	. .
Briquettes (dust)	12.5	MgO. 0.8	H_2O. 13.05	36.0	13.9	11.0	2.7	8.96	Insol. 45.6	0.66	5.0

furnace (Figure 1), a radical change was then inaugurated by making the hearth oblong, tapering very quickly toward the bridge, but less so toward the flue (Figure 2). The result was an enlarged hearth area served by the same grate area which, on account of the free development of the flame, could heat more effectively and smelt a larger amount of charge per unit of fuel consumption than with the old form. While at Argo, Colorado,[1] the reverberatory matting furnace of

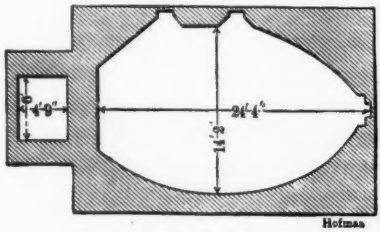

FIG. 1.

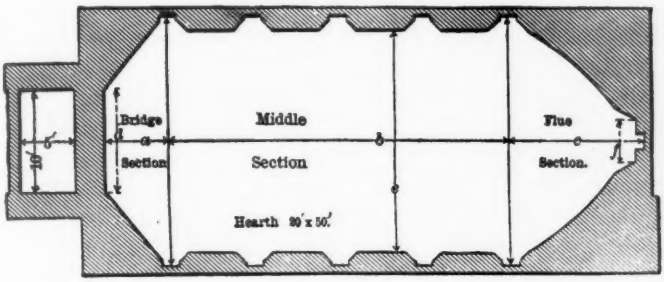

FIG. 2.

1878 had a hearth area of 105 square feet, with a ratio of hearth-to-grate area as 4.66 : 1; in 1894 the hearth area had been increased to 481 square feet and the ratio to grate area decreased to 15.03 : 1. At present the Montana furnaces reach in round figures a hearth area of 880 square feet, with hearths 50 feet long and 20 feet wide (in the middle section, Figure 2); the ratio of hearth-to-grate area differs, however, only slightly from the Argo standard of 1894. Such a furnace treats 105 tons of charge in twenty-four hours, producing 50-per cent. matte with a concentration of three to four into one. These large furnaces have many advantages over the smaller ones besides the greater

[1] Peters, *Modern Copper Smelting*, 1895, p. 445.

TABLE VIII.— Examples of Montana Reverberatory Furnace Smelting.

	A.	B.	C.	D.	E.	F.
Length of hearth	50 ft.	50 ft.	49 ft. 6 in.	50 ft.	42 ft. 6 in.	37 ft.
Length, *a*, of bridge section *	6 ft.	6 ft.	4 ft. 6 in.	6 ft.	42 ft. 6 in.†	. .
Length, *b*, of middle section *	28 ft.	32 ft.	32 ft.	32 ft.	42 ft. 6 in.†	16 ft.
Length, *c*, of flue section *	16 ft.	12 ft.	13 ft. 0 in.	12 ft.	42 ft. 6 in.†	. .
Width, *d*, of hearth at bridge *	10 ft.	10 ft.	10 ft. 9 in.	10 ft. 9 in.	15 ft. 9 in.	8 ft. 0 in.
Width, *e*, of hearth at middle *	20 ft.	20 ft.	20 ft. 2 in.	20 ft.	15 ft. 9 in.	14 ft. 6 in.
Width, *f*, of hearth at flue *	4 ft.	6 ft.	4 ft.	4 ft.	15 ft. 9 in.	5 ft. 0 in.
Hearth area, sq. ft.	842	886	878	878	688	. .
Hearth, thickness	22 in.	24 in.	19 in.	20 in.	28 in.	22 in.
Length of grate	10 ft.	10 ft.	7 ft.	10 ft. 9 in.	Producer gas, regenerative chambers	8 ft.
Width of grate	5 ft. 5½ in.	5 ft. 4 in.	5 ft. 6 in.	5 ft.		6 ft.
Depth of grate below top of bridge at bridge	1 ft. 10 in.	2 ft. 9½ in.	2 ft. 4 in.	2 ft. 8 in.		3 ft. 0 in.
Depth of grate at opposite end	1 ft. 2 in.	2 ft. 6½ in.	1 ft. 4 in.	2 ft. 2 in.		. .
Grate area, sq. ft.	53.9	53.3	53.0	53.75		48.0
Ratio, hearth : grate area	15.6	16.6	16.5	16.3		. .
Height of roof above bridge	2 ft. 7 in.	2 ft. 6½ in.	2 ft. 10 in.	2 ft. 8¾ in.	24 in.	2 ft. 0 in.
Height of roof above hearth at bridge	4 ft. 8 in.	4 ft. 4 in.	4 ft. 4 in.	4 ft. 8 in.	7 ft. 0 in.	4 ft. 10 in.
Height of roof above hearth at flue	2 ft. 1½ in.	2 ft. 10 in.	3 ft. 0 in.	2 ft. 2 in.	. .	2 ft. 4 in. to 2 ft. 8 in.
Width of bridge	3 ft. 0 in.	3 ft. 5¼ in.	3 ft. 0 in.	3 ft. 0 in.	4 ft. 1½ in.	. .
Size of flue at vulcatory	6 ft. x 30 in.	6 ft. x 30 in.	5 ft. x 30 in.	6 ft. x 30 in.	. .	36 in. x 6 ft. 6 in. to 7 in.
Size of flue leading to chimney	30 in. x 48 in.	30 in. x 30 in.	28 in. x 30 in.	30 in. x 30 in.	42 in. diam. at each end	30 in. x 36 in.
Chimney, inside diameter	5 ft. 6 in.	6 ft. 3 in.	6 ft. 0 in.	6 ft. 1 in.	. .	4 ft. 6 in. x 4 ft. 6 in.
Chimney, height	70 ft.	. .	70 ft.	75 ft. 3½ in.	. .	80 ft.

* See Fig 2. † The hearth has the form of a rectangle.

TABLE VIII.—Examples of Montana Reverberatory Furnace Smelting. (*Continued.*)

	A.	B.	C.	D.	E.	F.
Charge, weight, tons	25	15	24	18	35	13
Charge, time of melting, hours	5	3½	6½	4¾	6	5½
Charge, tons in twenty-four hours	112.5	105	90	90	150	70
Charge, tons per sq. ft. hearth in twenty-four hours	0.129	0.118	.102	0.102	0.218	. .
Rate of concentration	5.7 : 1	4.7 : 1	7.56 : 1	5 : 1	3½ : 1	5 : 1
Fuel, bituminous coal, manner of firing	Direct	Direct	Direct	Direct	Producer gas	Direct
Fuel, bituminous coal, per cent. ash	8.9	15.0	14.85	5.0	17.0-30.0	7-10
Fuel, bituminous coal, per cent. fixed carbon	51.9	44.5	44.5	55.0	45.0	50-52
Tons charge : 1 ton coal	3.05	3 : 1	4 : 1	2.81 : 1	2 : 1	2.5 : 1
Labor, in eight-hour shift*	2+2	2+2	2+2½	2¾+1½†	†	2½+1½
Matte, Cu	53.8	48.3	50	50	50	55
Ag ozs. per ton	45	35	100	66	20	. .
Au $ per ton	. .	. .	. .	2(?)	. .	. .
Sp. gr.	. .	4.8	. .	. .	4.77	. .
Slag, SiO_2	36.8	42.8	34.5	38.6	41.9	49
Fe(Mn)O	51.9	47.31	43.0	51.4	42.7	32-36
Al_2O_3	8.4	7.5-8.0	8.5	1.5	10.9	. .
ZnO	. .	. .	14.0	. .	. .	. .
CaO	1.1	1.2	. .	3.3	1.1	. .
Cu	0.75	0.40	0.60	0.40	0.58	0.65
Ag	0.45	0.30	1.0	0.40	. .	. .
Sp. gr.	. .	3.58	. .	. .	3.54	. .

* The first figure represents the regular crew, the second the auxiliary labor.
† Not comparable in this manner on account of gas producers.

capacity. The sides are less readily corroded by the slag (claying every twelve days versus every three days), as for a given amount of matte less surface is exposed to the slag; there is less variation in temperature on account of the greater reservoir of heat, which lengthens the life of the brick; the layer of slag is thinner, which allows quicker heating and causes less foaming when the charge is dropped from the hoppers; and the level of the matte remains more constant.

Fuel. — The fuel used is bituminous coal of various grades from Montana, Wyoming, and Utah, some of which are represented in Table IX.

TABLE IX.— Average Coal Analyses.

Further analyses of coals are given in the Appendix, Table XV.

Locality and State.	Name of Mine.	Kind of Coal.	Years Averaged.	H_2O Per cent.	Volatile Matter.	Fixed Carbon.	Ash.
Belt, Montana	Belt	Slack	1898-'99-1902	1.6	30.4	45.7	22.3
	Belt	Washed slack	1899-1900	0.8	33.0	49.2	17.0
	Belt	No. 1 nut	1900	0.2	38.1	47.2	14.5
	Belt	Pea	1900	0.8	33.4	48.9	16.9
	Belt	Washed pea	1900	0.9	35.3	50.5	13.3
	Belt	Washed nut	1901-1902	1.3	29.2	50.2	19.3
	Belt	Lump	1900-1901	1.5	29.3	50.8	18.4
	Belt	Lump	(?)	2.2	24.25	46.65	26.9
	Belt	Lump	(?)	2.4	22.1	43.2	32.2
	Belt	Mine run	1901	0.9	27.6	47.3	24.2
	Belt	Mine run	(?)	3.9	24.9	49.6	24.6*
Stockett, Montana	Sand Coulee	Slack	1898-1900	1.2	30.8	46.1	21.9
	Sand Coulee	Washed nut	1898-1901	1.4	31.8	50.4	16.4
	Sand Coulee	Mine run	1899	1.1	28.1	46.2	24.6
	Sand Coulee	Nut	1898-1900	1.1	30.3	45.8	22.8
	Sand Coulee	Lump	1900-1901	0.9	30.4	50.2	18.5
	Cottonwood	Slack	1898-1901	1.1	30.1	44.3	24.5
	Cottonwood	Nut	1898-1901	1.1	29.7	44.7	24.5
	Cottonwood	Lump	1899-1901	0.8	30.4	46.3	22.5
Sand Coulee, Montana	Lochray	Lump	1899-1901	1.6	30.2	47.3	20.9
	Lochray	Mine run	1899-1901	1.0	30.0	47.7	21.3
	Lochray			1.8	27.4	52.1	19.4
	Gerber	Lump	1900-1901	0.9	31.2	48.9	19.0
	Gerber	Mine run	1899	1.2	31.5	48.5	18.8
	Gerber			1.0	23.5	56.7	17.7
	Millard	Slack	1899	0.5	33.2	45.6	20.7
	Millard	Mine run	1899-1900	0.8	31.1	46.7	21.4
	Lewis	Slack	1898-1899	1.2	31.7	45.9	21.2
	Lewis	Lump	1898-1901	0.9	30.4	49.0	19.7
	Lewis	Mine run	1899-1900	1.0	30.5	46.6	21.9
Wyoming	Sweet Water	. .	1902	4.6	32.0	56.4	7.0
	Sweet Water	. .	1902	5.4	31.8	50.4	12.0
	Bear Creek		1902	7.85	36.45	46.65	9.05
	Diamondville	Slack	1902	2.6	37.58	45.12	14.70
	Diamond	Lump	1902	4.7	36.52	44.56	14.85
Utah	Gas coal†	. .	1902	4.8	33.0	59.1	1.9
British Columbia	Lethbridge	Slack	1898-1901	2.5	31.5	43.9	22.1
	Lethbridge	Nut	1898	4.3	31.6	45.0	18.8
	Lethbridge	Pea	1899	2.9	34.6	41.2	21.3
Pennsylvania	Anthracite	. .	1900-1901	0.5	7.8	80.6	11.1

* Average of ten months. † S 1.2 per cent.

Most furnaces are direct fired. Some furnaces are worked with natural draft, but the majority have undergrate blast. In one instance the original fire box has been changed to receive a Duff (Pittsburg) gas producer, with most satisfactory results as to saving of fuel and to shortening of the time required for melting. This latter is due to the fact that the ordinary grate has to be cleaned every four hours, while the grate of the Duff producer requires attention only once in twenty-four hours. On account of structural difficulties the producer had to be given up. In many furnaces air is admitted through the roof above the fire bridge in order to secure a better combustion and thereby save fuel. The air[1] travels through the vault underneath the furnace, rises some distance in two corners of the stack (square on the outside, circular on the inside), descends in the others, returns underneath the hearth through three or four horizontal channels on either side of the vault, rises in corresponding vertical boxes near the bridge, enters both sides of an air chamber on the roof above the bridge, and descends through slots into the fire-bridge flue. There is some doubt prevalent as to the efficiency of this superheating. With most furnaces the heat of the flame passing out of the furnace goes to waste. According to Hixon[2] attempts had been made to join the Brown-Allen-O'Hara roasting furnace to a reverberatory matting furnace, but without success, owing to mechanical difficulties. Hixon suggests improvements which are to overcome them. Lang[3] patented a combination of reverberatory matting furnace with a hand reverberatory roaster; at Kedabeg, in the Caucasus, such a combination is said to be[4] in successful operation. Attempts have been made by Thum[5] to combine the Belgian zinc-smelting furnace with a hand reverberatory roasting furnace for roasting blende, but without success. The main difficulty would seem to lie in the fact that, in order to roast successfully in a reverberatory furnace, it is essential to have a steady even temperature at the fire bridge, and with a reverberatory matting furnace, charged and discharged at intervals, this is not easily obtained, if it can be managed at all. In order to utilize the waste heat, one Montana plant has attached a 300-horse power Stirling boiler, Class A, $2\frac{1}{2}$, with 3,019

[1] See also Peters, *op. cit.*, p. 456.

[2] *Notes on Lead and Copper Smelting*, New York, 1897, p. 59.

[3] *Engineering and Mining Journal*, Aug. 6, 1898.

[4] *Berg- und Hüttenmännische Zeitung*, 1891, p. 449; *The Mineral Industry*, VI, p. 247.

[5] *Berg- und Hüttenmännische Zeitung*, 1874, p. 276.

square feet heating surface to one of its 50- by 20-foot furnaces. While an attempt of this kind in an Eastern refinery had proved a failure, owing to the diminished melting power caused by the reduced force of the draught, here it proved an unequivocal success. At first, when the boiler was set up in the usual way, the smelting power of the furnace was reduced to an unreasonable figure, which was restored by removing nearly all the baffle plates and thus giving the gases a straight course through the boiler.

One plant only has reverberatory matting furnaces with regenerative chambers similar to the open-hearth steel furnaces with stationary hearths. Tilting furnaces (hearths 13 by 16 feet with 176 square feet area), similar to the Campbell open-hearth steel furnace,[1] which were built in 1890 and 1892, were abandoned on account of their small capacity, and because of the fact that it was impossible, in pouring off the slag, to control the flow in such a manner as to prevent matte from passing over. Small amounts of matte in a large mass of slag cannot be distinguished by the eye. These furnaces were run, however, for several years as stationary furnaces, being gradually replaced by furnaces of larger size. The gas producer used is the Evans-Klepetko;[2] it is a modification of the well-known Taylor producer, is 15 feet high, has a 7-foot bosh and a throat 7 feet 4 inches in diameter. The points in which it differs from the Taylor producer are a revolving top, a mechanical feed discharging into a hopper in the centre of the top, a steam-jet reaching into the hopper, which, with a pressure equal to that of the gas in the furnace, prevents this gas rising through the coal; four water-cooled stirring-arms attached to the revolving top which reach down into the bosh and are so distributed that they cover the entire surface during one revolution; lastly, an attachment for the sluicing of ashes as they are discharged by the revolving grate. The producer gasifies from 10 to 12.5 tons coal in twenty-four hours, the top making one revolution in three minutes. The analyses in Table X give a comparison of the two kinds of coal used and of the ashes formed.

Details of Practice. — The coal is crushed through a $\frac{3}{4}$-inch ring and passed through a 5-millimeter screen; the oversize goes to the producers; the fines, forming about 30 per cent., are burned under the boilers. It will be seen that, owing to the high percentage of ash in the coal, the amount of fixed carbon retained by it is exceedingly high. The

[1] *Transactions American Institute of Mining Engineers*, XXII, p. 360.

[2] *United States Patent*, No. 704,527, July, 1902.

flue from the six producers serving one matting furnace has to be fired once in eight days to remove soot, tar, etc., and washed out once in six weeks to free it from ashes. An average analysis of producer gas taken from a large number of determinations gave CO_2, 9.4; CO, 16.8; CH_4, 2.9; H, 13.3; N, 57.6. The high percentage of carbon dioxide is probably due to the excessive use of steam. Bunte[1] has shown that the larger amount of steam that is admitted to a producer the larger is the volume of carbon dioxide formed. The gases, however, pass off from the producers with a temperature of 700° C., which would seem to counteract the cooling effect of the steam, and cause the carbon to burn rather to carbon monoxide than to dioxide.

The hearth of the regenerative matting furnace, 42 feet 6 inches by 15 feet 9 inches, has the same width at the ends as at the middle. While this arrangement gives a larger hearth area than in the furnace tapering at the ends, it has the disadvantage that the sides tend to bulge inwardly instead of outwardly, as is ordinarily the case. The furnace is built in five sections in order to allow for expansion, which amounts to 14 inches over all. The sections are separate in the roof; in the 18-inch side-walls the necessary spaces are left between the bricks. The upper tie rods run over 1 foot above the roof (from 9 to 12 inches thick), as this may rise as much as 9 inches; the lower tie rods, placed 4 feet below the brick hearth supporting the quartz bottom, were passed at first through 4-inch wrought-iron tubes; when these burned out the new tie rods were provided with a small central opening for water cooling. At present the rods are solid, $1\frac{1}{2}$ inches in diameter, and pass through a flue 2 feet 6 inches by 3 feet 8 inches, which is ample for air cooling. The checker work has a peculiar construction to permit cleaning, and thus reduce the slagging of the brick through dust. The flues are horizontal, 18 inches high by 7 inches wide by 11 to 21 feet long, according to position; they are constructed of fire brick. The 9-inch brick forming the tops and bottoms are laid $4\frac{1}{2}$ inches apart, leaving open spaces to allow the dust to fall and collect in a pit at the bottom, from 4 to 5 feet deep. The heating surface is not very much diminished by this arrangement, as the sides of the horizontal bricks are exposed to the hot gases; this would not be the case if the bottoms and tops did not have the open spaces. The dimensions of the air chambers are: width 8 feet 6 inches by length 14 feet $7\frac{1}{2}$ inches by height 13 feet 9 inches; of the gas chambers, width 7 feet by length

[1] Ledebur, *Gasfeuerungen*, Leipsic, 1891, p. 34.

TABLE X.—Analyses of Producer Coal and Ash.

Mine Run of the A. C. M. Company Belt, Montana.

Coal.*					Ash.†			
Total H_2O.	Comb. H_2O.	Volatile Matter.	Fixed Carbon.	Ash.	H_2O.	Volatile Matter.	Fixed Carbon.	Ash.
4.0	1.0	23.6	51.3	24.1	1.1	10.4	45.4	43.1
4.7	1.8	21.4	48.4	28.4	1.4	3.3	34.3	61.0
4.4	1.4	24.4	49.3	24.9	1.5	4.3	30.4	63.0
4.1	1.4	25.4	48.8	24.4	0.9	12.3	40.5	46.3
4.2	1.4	23.9	46.9	27.8	1.6	4.6	36.3	57.5
4.1	1.4	25.5	49.9	33.2	1 2	9.6	24.1	70.1
Mine Run of Lochray Mine, Sand Coulee, Montana.*								
5.0	0.9	27.5	51.7	19.9	0.3	2.4	36.4	60.9
4.4	1.1	26.3	52.6	20.0	0.2	3.0	13.9	82.9
4.5	0.8	27.0	50.7	21.5	0.3	3.2	24.3	72.2
3.8	0.7	27.3	52.7	19.5	0.3	3.8	24.1	71.8
Mine Run of Gerber and Lochray Mines.								
					0.3	7.0	32.5	60.3

* For heat values of these and of other coals, see Appendix, Table XV.

† Not to be depended upon too much on account of difficulty of obtaining a good average sample.

The Anaconda coal alone does not work well in the producer; it is added to the Lochray coal, which alone gives very satisfactory results.

The ash from the Gerber-Lochray coal, freed from carbon, shows the following composition:

SiO_2, 44.15; Fe_2O_3, 16.6; Al_2O_3, 37.9; CaO, 1.0; MgO, 0.4; S, 0.04.

14 feet $7\frac{1}{2}$ inches by height 13 feet 9 inches. The heating surfaces, including exposed walls, are 4,900 and 4,200 square feet, respectively. While long checkers save fuel, they tend to shorten campaigns; the reverse is the case with the short checkers. Air enters the hearth through a single port 15 feet 9 inches by 2 feet; the gas through four ports $20\frac{1}{4}$ by 30 inches. The products of combustion from the furnace descend in a chamber at one end of the checkers, and after passing through these, ascend in the other at the opposite end. The life of the

checkers is about three months. In order to lengthen it, experiments are under way to have the gases go, during the charging period, through a by-pass direct into the main flue, and thus prevent the inevitable dust from settling in the checker flues.

The working-bottoms of the furnaces, as shown in the table, have a thickness ranging from 19 to 28 inches at the lowest point, the tap-hole. Both single- and double-bottoms are found. The single bottom is more common with ore smelting, as here the furnace has to carry a relatively light material in comparison with the refining furnace with which the double bottom is more prevalent. The bottom-sand is a sugar-like quartzite, with 95 per cent. silica, the rest being alumina and ferric oxide. It is crushed between rolls to bean-size, say through a $\frac{1}{4}$-inch ring; 80 per cent. of it will pass through a 5-millimeter screen. Two examples of putting in a bottom will show how great are the differences in practice. One furnace has a hearth 20 by 50 feet, is direct-fired, and takes 73 tons of crushed sandstone. Supposing the furnace to be dry and warm, from 6 to 7 tons of sand are charged and heated for 6 hours, being rabbled at intervals. These operations are continued until one-half of the total sand required has been introduced, when the whole is heated to the fritting of the surface. The other half of the sand is now charged in the same manner as was the first, and brought to fritting. Converter slag is now poured in to cement the particles of sand more firmly together; from three to four charges of calcines are worked, and the furnace tapped dry after the fourth charge. The furnace is allowed to cool for three to four hours, and is ready for regular work. The time required for putting in the bottom is four days. The bottom under consideration was put in in February, 1900, and shows little wear today (June, 1902).

The other extreme is represented by a gas-fired furnace with regenerative chambers; the rectangular hearth is 42 feet 6 inches by 15 feet 9 inches. It is presupposed that the furnace is new and requires drying and warming. A light wood fire is started on the hearth in half a dozen places; the valves are reversed every half hour. In ten days the furnace will be at a dull red; in two days more it will be hot enough to ignite producer gas. The gas is turned on in small amount at first; after three days the wood ashes are taken out; on the fourth or fifth day (the sixteenth or seventeenth after starting) the furnace will be a bright cherry red. Five tons sand are charged through the hoppers in the roof and spread; they cover the hearth to a thickness of

2 inches. The sand is calcined, turned over, and the furnace brought to a normal heat; charging of sand and heating it are continued in five- to six-hour intervals until about 60 tons have been introduced, care being taken to have the sand thoroughly calcined. The furnace is now brought in from thirty-six to forty-eight hours to the highest heat that can be obtained without endangering the brickwork, and the sand brought to a set which amounts to about 3 inches in thickness. The high heat is maintained for sixteen hours, and the furnace then cooled down in eight hours to a dull cherry red. Enough calcines (giving 45-per cent. copper matte) are dropped through the roof to cover the hearth to a depth of 3 inches; they are melted down and are absorbed by the sand bottom. The furnace is cooled as before, and a second charge of calcines given and smelted. Part only of the charge will be taken up by the hearth; the rest is tapped out. The furnace is again cooled, from 12 to 15 tons ore-charge are given, melted down, and the furnace tapped dry and cooled to a dark red, but more slowly than before. It is again fired up, worked for three days as under normal conditions, and then tapped dry and allowed to cool to a cherry red. It is now safe for any kind of work. The time required to bring a new, cold furnace to this point is three weeks. The object of the repeated coolings of the furnace is to harden the bottom. A bottom put down without cooling is liable to be mushy when it is hot, even though it be hard when relatively cool.

The ore-charges consist mainly of roasted concentrates, with fines from first-class ore; sometimes small additions of limestone are made. In converter plants low-grade blast-furnace matte is often poured into the reverberatory matting furnace in order to raise the percentage of copper.

In regular work the charge is collected above the furnace in from five to seven hoppers, having discharge pipes with gates, and dropped onto the hearth. With non-regenerative furnaces the bulk of the charge is collected near the fire bridge; thus with five hoppers, *e.g.*, seven and one-half volumes of the charge will be dropped through hopper Number 1 near the fire bridge, then six and one-half volumes through hopper Number 2, five through Number 3, three through Number 4, and one through Number 5 near the flue bridge. With a regenerative furnace having, for example, seven hoppers, charge will be first dropped through the end hoppers that it may flow toward the centre, then through the central hopper and its two neighbors, and

lastly through the two remaining ones, the flow being regulated with the gates that there may be an even distribution. The charge is now melted down and stirred at intervals in order to break up lumps. With a furnace that has been tapped dry, the bottom has to be rabbled in order to bring adhering melted parts to the surface. When melted and thoroughly liquefied the slag is skimmed into a settling box or a Devereux-Livingstone settling pot,[1] that particles of matte carried out by the slag may be collected and saved. With zinckose ore it is important to give a good fire before tapping, that there may be a good separation of the mushy, zincky matte (which, being light, floats on the surface of the heavy matte) and the slag. The overflow-slag is collected either in waste-slag pots of from 5 to 6 tons capacity, hauled to the dump electrically and poured, or it is granulated and carried off by a stream of water. Granulating slag from the reverberatory matting furnaces was first introduced in 1888 at the Anaconda Lower Works. In order to reduce the copper contents of this slag, fine pyrites has been sprinkled into it before the final firing up, with the hope that slagged copper may be sulphurized and carried into the matte. The matte is removed from the furnace wholly or only in part, depending on whether it is to be collected and shipped or transferred to the converters. When the furnace is to be tapped dry, which occurs only after several charges (with high concentrations as many as twelve charges) have been treated, and the hearth holds from 18 to 20 inches of matte, this is run into sand moulds or cast-iron moulds of similar form, or it is first collected in a steel ladle, and then poured into the form of flat cakes. With a converter plant only part of the matte is removed after every charge, the aim being to keep the level as nearly constant as possible. With furnaces that are tapped dry, the sides are patched or fettled after every tap. With converter plants, fettling of the matting reverberatories is resorted to only once every four or five weeks. As fettling material, siliceous gold ore has, in some instances, replaced the usual sand. A furnace runs about ninety days before it needs sufficient repairs to require shutting down. The subjoined partial analysis of flue dust collected in the gas checkers of a furnace gave:

Insoluble, 91.1 per cent.; SiO_2, 62.2; Fe_2O_3, 8.2; Al_2O_3, 25.7; CaO, 1.8; MgO, none; S, 0.30; CuO, 2.3.

Evidently considerable coal ash was carried over by the gas.

[1] Hofman, *Lead*, 1899, p. 267.

The material formed in the air checkers resembles slag very much in analysis, except that it is higher in copper (*i. e.*, 5 to 9 per cent.), silica, and alumina, and a trifle lower in iron.

VII. Converting Copper Matte.

The converting of copper matte has undergone many changes in the form of apparatus and the mode of working since 1884, when the first converter was put into operation.

The original Parrot converter,[1] swinging in a vertical plane, was $8\frac{1}{2}$ feet high and 5 feet in diameter, outside dimensions. The matte was melted down in a cupola and tapped into the converter through a launder. In 1886 the original method of working in separate stages was changed, and 50-per cent. matte was brought forward to coarse copper in one operation. As the initial charge was 2,500 pounds of matte, increasing to a maximum of 9,000 pounds, the siliceous lining being eaten and worn away, the amount of coarse copper produced in a blow was relatively small. The largest upright converter in Montana[2] is 14 feet $2\frac{1}{4}$ inches in height and 7 feet in diameter, outside dimensions. It works with an average charge of 5 tons and a final charge of 11 tons of 50-per cent. copper matte, receiving it as direct matte from the forehearth of the blast furnace or from the reverberatory furnace. Working direct matte was planned in 1890–91 by the late C. O. Parsons, and carried out by F. Klepetko in 1892; matte was transferred from reverberatory and blast furnaces by means of ladle and electric crane to the converter. At two plants a medium-sized converter was lifted by means of an overhead traveling crane from its trunnion supports and brought to the blast or reverberatory matting furnace to receive its liquid charge; this method has become obsolete, but it has its advantages.

At present the Parsons-Klepetko method is generally employed. A converter charge is tapped from the blast-furnace forehearth, or the reverberatory matting furnace, into a steel ladle managed by an electric traveling crane and poured into the converter. Ladles vary in size; a common form has a diameter of 5 feet and a height of 3 feet 10 inches. The lining, ordinary loam, is plastered on by hand,

[1] Peters, *op. cit.*, p. 529. Stickney, *The Mineral Industry*, I, p. 154.

[2] The largest upright converters are those of Aguas Calientes, Mexico: height, 16 feet; diameter, 8 feet.

dried by an air current and then by a fire (wood, coke); converter slag, to the thickness of 3 inches, is then poured in. Matte-ladles are used also as slag-ladles. A lining is good for $9\frac{1}{2}$ tons of copper.

When the 50-per cent. matte has been blown to white metal, the slag is skimmed and the converter very often recharged with 50-per cent. matte and blown through the slagging stage; but more commonly the original charge of matte is blown to coarse copper after skimming without addition of fresh matte. Doubling charges is a common procedure on matte running less than 50 per cent. copper. After skimming the slag the white metal is brought forward to coarse copper. By these double charges considerable amount of copper is produced in one blow, to be poured into the form of shipping ingots, from 200 to 250 pounds, or anodes, or into a refining furnace, to be refined before casting into anodes.

The upright converter has been replaced in the new plants by the David-Manhès or barrel or trough converter, swinging in a horizontal plane. Here we find two sizes: the smaller, in more general use, is 7 feet in diameter and 10 feet 6 inches long; it receives an initial double charge of 5 tons and a final double charge of 12 tons 50-per cent. matte; the larger one is 8 feet in diameter and 12 feet 5 inches long; the initial double charge is 10 tons, and the final double charge is 20 tons. The main reason for replacing the upright by the horizontal converter is the low blast pressure required by the latter type, 10 versus 15 pounds, although this advantage suffers greatly by the difficulty of making the lining firm, a thing relatively easily accomplished in the vertical converter. Converting matte in a bottom-blown upright vessel has been tried. This was 13 feet high and 7 feet in diameter, had sixteen $\frac{3}{4}$-inch tuyères placed in a manner similar to that which obtains in making steel by the Bessemer process. The experiments were successful so far as the process proper was concerned, *i.e.*, 50-per cent. matte was satisfactorily brought forward to coarse copper, which remained sufficiently fluid to be poured satisfactorily. They appear to show that the failure of Manhès to use a bottom-blast converter[1] in his first experiments was probably due to his operating with too small a vessel from which matte and slag were blown out, and with too little matte, which furnished only a small amount of copper quickly chilled by the blast. The Montana experiments have not been followed up for the present, as the life of the bottom was too short, parts of it floating up

[1] Grüner, *Bulletin de l'Industrie Minérale de Ste. Étienne*, 1885, XIV, p. 607.

during a blow. The experience gained so far tends to show that the bottom-blown converter works more quickly than the side-blown, that the wear of the lining is more even, and that the critical points are less easily discovered and have to be watched with great care. When the question of lining the bottom will have been settled satisfactorily, it is believed that blowing from the side will be given up, at least with a large converter having an initial charge of 5 tons 50-per cent. matte. A gyratory motion of the charge, obtained by placing the bottom tuyères inclined, as in the David *Selecteur*,[1] may lengthen the life of a converter lining.

In all plants the converters are placed in a single row, forming a straight line. Back of them is the main flue (vacuum 0.1 inch water), with projecting hoods, into which the converters discharge their gases, vapors, and finely divided particles of charge (called flue-dust). The main flue ends in a dust chamber (temperature 370° C.) connected with a stack. At some works the matte is poured into a converter at the front from a suspended ladle; at others it arrives at the back (above the main flue) in a ladle, and is poured into a launder, which delivers it at the front. A stall is served by from two to two and one-half converters and from one and one-half to two ladles. The lining of converters, both of body and of caps, is still causing trouble, as in the most favorable case the body has to be removed after 28 tons of copper have been treated; the cap has a longer life. The material is, of course, siliceous. A greenish granular sandstone (SiO_2, 64.4; FeO, 7.6; Al_2O_3, 16.5; CaO, 1.1; MgO, 1.8; Ignition loss, 4.7), which readily crumbles when exposed to the air, is used without any admixture of a bond. A crushed quartzite pugged with 5 per cent. clay gives a satisfactory mixture (SiO_2, 88.6; Fe_2O_3, 3.1; Al_2O_3, 4.0; CaO, 1.6); lining-sand has been mixed with slimes from the ore-dressing works (insoluble, 75.3; SiO_2, 60.0; Fe, 2.2; Al_2O_3, 19.9; S, 6.8; As and Sb, 0.75; Cu, 6.05) and with second-class ore (insoluble, 68.9; SiO_2, 57.0; Fe, 9.6; Al_2O_3, 10.4; S, 15.95; Cu, 5.95; Ag, 2 ounces per ton), giving the following material:

Insoluble, 82.2; SiO_2, 63.0; Fe, 5.9; Al_2O_3, 17.3; CaO, 1.5; MgO, 1.3; S, 2.1; Cu, 1.0; H_2O, 11.0; ignition loss, 4.1.

Another mixture for the body of the converters consists of quartzite (SiO_2, 90 to 91; Al_2O_3, 2.0; FeO, 3.0; Ag, 10 to 20 ounces per ton)

[1] P. L. Burthe, *Annales des Mines*, 1898, XIII, p. 621. *Engineering and Mining Journal*, Oct. 22, 1898.

and slimes (SiO_2, 60; Fe, 4 to 5; Al_2O_3, 15; S, 9; Cu, 5 to 7) to form a lining (SiO_2, 85; Fe, 3.3; Al_2O_3, 4; S, 3; Cu, 4; Ag, 12 ounces per ton).

Table XI gives analyses of all the materials used to make the converter-lining.

TABLE XI.—ANALYSES OF CONVERTER LININGS.

	SiO_2.	Fe.	Al_2O_3.	CaO.	MgO.	Loss on Ignition.	S.	Cu.	Oz. Ag. per Ton.	As and Sb.	H_2O.
Greenish, granular sandstone, Great Falls	64.4	5.9	16.5	1.1	1.8	4.7	. .	. .	. .	. .	. .
Crushed quartzite, with 5 per cent. clay	88.6	2.2	4.0	1.6	. .	. .	. .	. .	. .	. .	. .
Sandstone, slimes and second-class ore	68.0	5.9	17.3	1.5	1.3	4.1	2.1	1.0	. .	. .	11.0
Quartzite and slimes	85.0	2.5	4.0	. .	. .	. .	3.0	4.0	12.0	. .	. .
Sandstone	64.4	5.9	16.5	1.1	1.8	4.7	. .	. .	. .	. .	. .
Slimes	60.0	2.2	19.9	. .	. .	. .	6.8	6.05	. .	0.75	. .
Second-class ore	57.0	9.6	10.4	. .	. .	. .	15.95	5.95	2.0	. .	. .
Quartzite	90-91	2.7	3-2	. .	. .	. .	. .	. .	10-20	. .	. .
Slimes	60.0	4-5	15	. .	. .	. .	9.0	5.7	. .	. .	. .

In some instances siliceous gold ores have been worked in as a lining material. The material for the cap is generally made more plastic than that of the body. The lining usually consists of body-lining material and 5 to 10 per cent. of clay, and is mixed with more water than the body material. If the lining of the body were eaten away solely by the scorifying action of the iron, its life would be much longer than it is. The chemical action in some instances plays even a subordinate part to the mechanical wear by the swash of the liquid charge, which causes part of the lining to break off. This is especially observable in the barrel converter. The chemical wear takes place mainly near the tuyères, the mechanical wear at the opposite side near the top of the body. In an upright converter 14 feet 2 inches high and 7 feet in diameter, the bottom lining will be 22 inches thick, the side lining at the tuyères will be 30 inches thick, and on the opposite only 18 inches, while at the top of the body the lining on the tuyère side will be $17\frac{3}{4}$ and opposite $27\frac{3}{4}$ inches thick.

In a horizontal converter 10 feet 6 inches long and 7 feet in diameter, the bottom will be 18 inches thick, consisting of a 4-inch segmental layer of mixture tamped onto the steel shell, followed by a slightly curved $4\frac{1}{2}$-inch course of brick and by a 10-inch layer of mixture. The sides are lined with a course of $4\frac{1}{2}$-inch fire brick, special brick $6\frac{1}{2}$ by $6\frac{1}{2}$ by 4 inches being used at the tuyère level. A cross section through the tuyère level will show the sand lining to be 30 inches at the tuyères and 16 inches opposite; at the top of the body the corresponding dimensions will be 20 and 20 inches. The longitudinal section will show the side lining to be 16 inches at top of the bottom lining and 34 inches at the top of the body. The ends are not lined with brick. Sometimes the lining is still rammed in by hand; an air-drill adapted for the purpose or the flask-tamper of the foundryman is more common.

The ramming is done by contract, a premium being paid if the converter makes more than the standard amount of copper on a lining, and deductions made if it falls below it. If the lining is to be patched, the vessel is first emptied and then cooled with water from the outside; then lining material is rammed in or a few scoopfuls of infusible odds and ends are thrown in, tamped down, and slag thrown over the patch. Occasionally matte is run into the empty converter and allowed to solidify. Experiments in lengthening the life of the converter-lining by pouring siliceous blast-furnace slag into the converters, with the hope that the excess of silica over that of the normal converter slag would combine with the oxidized iron from the matte, have been only partially successful.

The difficulty and expense of handling converter slag have been overcome in one plant by the use of slag casting-machine.

The average time to complete a charge with a 5-ton upright converter is two hours fifty-five minutes; this figure is reduced to two hours twenty minutes, if the time taken up by pouring and waiting is excluded. The blowing of a single charge takes forty-seven minutes; a double charge takes eighty-two minutes. In pouring the slag a rabble is pushed in and out of the stream of slag. The appearance of bright, sizzling specks is a sign that white metal is coming over. A few partial analyses of the products obtained in converting are subjoined:

TABLE XII.— ANALYSES OF CONVERTER PRODUCTS.

	Cu.	SiO_2.	FeO.	Al_2O_3.	CaO.	Ag, Au, Ozs. per Ton.	S.	As and Sb.
Slag	2.0	30.0	55.2	9.5	1.0	. .	. .	. .
Slag	1.6	29.8	57.8	10.0	1.0	0.8	. .	. .
Slag	5-6	31-32	54.0	. .	. .	. .	. .	. .
Dust	37.8	1.5	7.4	1.9	None.	. .	14.3	4.8
Dust	63.4	3.8	12.8	. .	. .	53.8	16.0	. .
Dust	65.0	. .	. .	. .	. .	. .	. .	. .
Pig copper	99.0	. .	. .	. .	. .	9-40	. .	. .

The labor required per shift for a single converter is made up of one skimmer, one puncher, one helper, and one sampler.

With three large horizontal converters running as steadily as possible, the number is one skimmer, two punchers, two helpers, and two samplers.

In Table XIII some of the data given previously have been summarized, and some additional facts have been added.

TABLE XIII. — MONTANA CONVERTERS.

Converter.	Grade of Matte.	Horizontal Section at Tuyères.	*Lining Material per Converter.	Lining per Ton Copper Resulting.	Copper per Converter.	Tuyères.		Maximum Pressure.
	p. c. Cu.	Sq. ft.	Tons.	Tons.	Tons.	No.	Diam.	Lbs.
Upright vessel, 7 feet x 14 feet 7 inches	51.1	7.07	7.63	.53	14.8	8	$\frac{3}{4}$	18
Medium barrel, 7 feet x 10 feet 6 inches	51.7	6.66	8.40	.32	26.2	14	$\frac{3}{4}$	12
Large barrel, 8 feet x 12 feet 6 inches	45.0	17.79	16.60	.80	20.0	18	$1\frac{1}{4}$	15

* These data do not show the amount of lining material required to line a *new* converter, but only the average weight put in after a lining is worn down to the point where further charges would endanger the iron shell.

It will be seen that the medium barrel uses much less lining per ton of resulting copper than the upright vessel, although the grade of the matte differs only slightly. This may be explained by the more siliceous character of the lining used, as, except for the mechanical effect of making the material plastic, alumina would be of no service in scorifying; furthermore, the lining of the upright vessel runs higher in iron.

The lining of the large barrel is of about the same quality as that used with the medium barrel; but as the matte of the former is of a lower grade, the quantity of lining used per ton of resulting copper is naturally higher.

Converter Lining.	SiO_2.	Fe_2O_3.	Al_2O_3.	CaO.	MgO.
Medium barrel	91.0	3.0	3.8	1.6	. .
Upright vessel	64.4	8.4	16.5	1.1	1.8

In the upright vessel the amount of free air used per ton of copper converted is about 200,000 cubic feet, not allowing for "leakage" in the blowing engines.

Yield of Metal. — The general yield of metal in roasting, smelting, and converting, in treating an ore assaying 10 to 12 per cent. copper and about 4 ounces silver per ton, is copper 92 + and silver 95 + per cent.

VIII. Electrolytic Refining of Copper.

The work of the two electrolytic refining plants in the Butte district, at Great Falls and Anaconda, is of especial interest, as they work under different conditions of power, while they treat similar anode copper by the multiple process. The works at Great Falls, driven by water power, have 312 tanks, 9 feet 9 inches long by 2 feet 4 inches wide by 3 feet 9 inches deep, in three sets of 104 tanks, each set having a separate solution; a tank is charged with twenty anodes and twenty cathodes; the current is of forty amperes per square foot of cathode area; two Westinghouse direct-current machines generate 9,000 amperes at 200 volts. The works of Anaconda,[1] driven by steam power, have 1,400 tanks 8 feet 2 inches long by 4 feet 5 inches wide by 4 feet 6 inches deep in sets of 200 each. A tank is charged with thirty-eight double anodes and eighty cathodes; the current is of ten amperes per square foot of cathode area. There are seven dynamos, each generating 4,000 amperes at sixty volts for 200 vats.

The anodes of the Butte district assay Cu, 98 to 99 per cent.; Ag, 40 to 120 ounces per ton; Au, 0.1 to 1.5 ounces per ton. They are cast either direct from the converter, or the converter copper is first refined in a reverberatory furnace.[2]

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[1] *Engineering and Mining Journal*, Sept. 19, 1896.

[2] Refining converter copper before casting into anodes has been given up at Great Falls, in spite of the advantages mentioned later on in the text as regards corroding, because the gain did not warrant the expense. At Anaconda, where power is generated by steam, it is of considerable advantage to have refined anodes.

The converter charge is blown to the point at which the copper still retains considerable sulphur dioxide in solution. In pouring anodes from the converter, the copper strikes a launder swung in position, which delivers the copper into open flat moulds resting on a car moved slowly on rails laid between the posts of a stand. The strong evolution of sulphurous gas upon cooling makes the anodes very uneven. The moulds are of copper, but have cast-iron centres that take up the impinging force of the stream of metal. If the copper is first to be refined, the metal from several converters is emptied into ladles which discharge their contents into the stationary refining furnaces. Here the rabbling of the copper, in order to cause the oxidation necessary to produce set copper, has been replaced by forcing air into the copper by 1-inch wrought-iron pipes inserted through the side- and end-doors of the refining furnace. The two hours required for rabbling have been reduced to one and one-quarter hours by blowing; 6 feet of pipe are used up with a 40-ton charge. Oxidation is continued until a small ladle-sample when broken shows the characteristic single bubble in the depressed centre. Care is taken to bring the copper only to this point, when the copper is saturated with cuprous oxide, as otherwise the surface will quickly become oxidized and covered with melted oxide. The set copper is then poled to tough-pitch copper, fifteen poles (3 inches at end, 7 inches at butt, and 15 to 18 feet long) being required for 50 tons of copper. From the reverberatory furnace the anodes are cast by means of an improved form of Walker casting machine.[1]

A tilting reverberatory furnace constructed along the general lines of the Brückner cylinder was used at the old Anaconda works, but given up after a short trial.

Converter anodes are less evenly corroded, give more scrap, and furnish a mud of lower grade than do anodes from the refining furnace. Coarse copper, *e.g.*, with Cu, 99.27 per cent.; As and Sb, 0.07 per cent.; Ag, 61.14 and Au, 0.22 ounces per ton, cast from the converters into anodes gave 8 per cent. scrap, and the anode mud, after having been passed through a 40-mesh sieve, washed and dried, contained Cu, 41 per cent.; Ag, 7,000 and Au, 18 ounces per ton. Similar copper refined in the reverberatory furnace and then cast into anodes gave in corroding only 5.5 per cent. scrap, while the 40-mesh anode mud assayed Cu, 18 per cent.; Ag, 15,000, and Au, 38 ounces per ton.

[1] *The Mineral Industry*, VII, p. 252.

The Great Falls anodes are 24 inches wide by 35 inches long by 3 inches thick at the top and $2\frac{1}{2}$ inches thick at the bottom; they are beveled at the bottom; the two lugs on top are $4\frac{1}{2}$ inches wide by $2\frac{1}{2}$ inches long by 2 inches thick. The anodes are suspended from cast-copper crossbars ($1\frac{1}{2}$ inches by $1\frac{1}{4}$ inches) by means of the Morrow clip,[1] a loop of sheet copper $2\frac{1}{8}$ inches wide (recently replaced by a rod $\frac{1}{4}$ inch in diameter) by $3\frac{1}{2}$ inches high, the ends of which are allowed to project into the anode moulds before they are filled with liquid copper. A converter anode weighs 500 pounds, and is sufficiently corroded in about eighteen days to be withdrawn and regarded as scrap, a new anode being put in its place. The Anaconda anodes are 24 to $24\frac{1}{2}$ inches wide, $32\frac{7}{8}$ inches long, and $1\frac{1}{4}$ inches thick at the top, and 1 inch at the bottom. They have two top lugs 6 inches wide with holes, and are suspended in pairs from iron copper-covered crossbars ($\frac{1}{2}$ by $1\frac{3}{4}$ inches), nineteen to a tank, by means of copper hooks $\frac{3}{8}$ inch thick. An anode weighs 230 pounds and is corroded in thirty-seven days.

The cathodes are made in the usual way in tanks set apart for the purpose. They are slightly longer and wider than the anodes. At Great Falls they are 26 inches wide by $36\frac{1}{2}$ inches long and weigh $2\frac{1}{2}$ pounds; at Anaconda they are 11 inches wide by 33 inches long and weigh $\frac{3}{4}$ to 1 pound. At Great Falls the cathodes are suspended by the Morrow clip, which is fastened to the starting sheet by a machine,[2] which punches a hole through the loop and sheet, bends over the lugs, and clamps them fast. At Anaconda the upper rim of the starting sheet is bent and clamped over the ends of a piece of sheet copper $4\frac{1}{2}$ inches wide by 11 inches long; four cathodes go to a crossbar; the electrode-distance ranges from $1\frac{1}{2}$ inches to 2 inches.

The tanks are made of 3-inch plank, and are lined with 8-pound lead; the bottoms have a board cover to protect the lead lining from pieces of anode copper that may become detached. At Great Falls the vats are arranged in pairs with three conductor bars, the central bar serving as cathode-bar for one tank, and as anode-bar for the other.[3] At Anaconda, with the Thofern disposition, the tanks are arranged in single rows.[4]

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[1] *United States Patents*, No. 621,121, March 14, 1899; No. 631,471, Aug. 22, 1899.

[2] *United States Patent*, No. 600,498, March 8, 1898.

[3] Illustration in *The Mineral Industry*, VII, p. 186.

[4] *Berg- und Hüttenmännische Zeitung*, 1893, p. 54.

The electrolyte contains with the heavier current 170 grammes concentrated sulphuric acid and 42 grammes copper per litre; with the lighter current, 150 grammes acid and 40 grammes copper. The resistance appears to begin to increase when the copper rises above 42 grammes. The temperature of the electrolyte at the head tank is 64° C. and 50° C., and the circulation six and three gallons per minute. Solutions are admitted and drawn off either at the ends or the sides, precautions being taken by means of perforated lead baffle-plates to have an even distribution, and thus avoid stirring up any mud. The different forms of plunger pumps, which were formerly used to raise the electrolyte, have given way to Pohle's air-lift pumps working under a pressure of 25 pounds. These have proved very satisfactory, especially since the delivery pipe has been cut in two near the discharge and the ends joined by a rubber hose, thus breaking any electric connection. The electrolyte becomes enriched in copper, and takes up iron, arsenic, and antimony. A foul solution contained, *e.g.*, per litre:

Cu, 51.8 grammes; Fe, 13.2 grammes; As, 14.02 grammes; Sb, 0.62 grammes; H_2SO_4, 48 grammes.

The low percentage of antimony is due to the practice of adding daily enough concentrated crude hydrochloric acid to the head tank to maintain .04 grammes chlorine per litre in solution. This addition precipitates antimony, it is believed, as oxychloride. When there is deficiency of hydrochloric acid, a sample-plate becomes streaked, tarnished, black, and at the same time brittle. The excess of copper over the normal is removed by shunting off part of the electrolyte and making it pass through tanks with lead anodes. By retarding the flow of this current through a second set of tanks, arsenic will be deposited with the rest of the copper. The slime deposited in such a purifying vat showed Cu, 57.1 per cent.; H_2SO_4, 11.8 per cent.; Ag, 0.3 per cent.; As, 7.2 per cent.; Sb, 1.8 per cent.; H_2O, 18.4 per cent.; difference: traces of $PbSO_4$ and traces of Fe.

While the difference in potential between electrodes with soluble anodes is 0.6 (Great Falls) and 0.3 (Anaconda) volt, with insoluble anodes it rises to 2.5 and 2.0 volts. When iron is to be removed the solution, freed from copper and arsenic, is concentrated by heating to one-third of its volume and then cooled, when most of the iron will crystallize out in the form of ferrous sulphate. At Great Falls the cathodes are renewed every second day. They weigh 55 pounds and

contain Cu, 99.95 per cent.; As, 0.0012 per cent.; Sb, 0.0033 per cent.; Ag, 1 ounce per ton. Although this method of working uses up a large number of starting-sheets, it is found to be advantageous, as with the large current of forty amperes there is a loss in efficiency on account of short circuiting if the cathodes are allowed to grow heavier. Thus with two-day cathodes there is an ampere efficiency of 91 per cent., while with four-day cathodes this falls off to 85 per cent. The cathodes, taken out four at a time by means of an overhead traveling pulley, are transferred to lead-lined cars and replaced by starting-sheets. The twenty cathodes in a car are then raised together by means of an electric crane dipped into water, drained, dipped into milk of lime, and allowed to dry. The coat of lime protects the plates in melting down from the sulphurous gases in the products of combustion of the refining furnace.

At Anaconda cathodes are removed only when the anodes are completely corroded. After two days' depositing they are, however, taken out and straightened. A tank receives at once all its anodes; in the same way all the cathodes are taken out by one operation.

The anode mud made at Great Falls is removed by a steam-injector pump and rubber hose from the electrolyzing vat into a settling tank covered with a forty-mesh screen; it is then drawn off into an acid egg, forced through a filter press, steam dried (when it still retains 2 per cent. water), broken up, sampled, sacked and sold.

At Anaconda the anode mud is discharged through an opening in the bottom of the electrolyzing vat, after the solution has been drawn off from the side, into a V-shaped trough leading into a tank; from this it is drawn into an acid egg, and forced into a tank in the refining department, drawn onto filter cloths, washed, dried, sampled, sacked and sold. Until recently the mud was treated at the works. It was melted down in a reverberatory furnace, refined by the addition of nitre, cast into small ingots, and parted with sulphuric acid, the silver being precipitated by means of copper.

The anode scrap at Great Falls is remelted in the converter, while at Anaconda it goes back into the anode furnace.

The leading facts of the two refining plants are brought together in Table XIV.

Refining of Cathodes. — The reverberatory furnaces used for refining converter and cathode metal have retained the pear-shaped form of the original Welsh furnaces. They have charging doors on both sides,

and the usual working and ladling doors at one end; and there are air flues in the bridge leading onto the hearth.

The following are the leading details of a standard furnace: length of hearth, 22 feet 6 inches; width at bridge, 7 feet 1.5 inches; at middle, 14 feet 8 inches; at flue, 1 foot 8 inches; hearth area, 225 square feet; thickness of hearth at bridge, 2 feet 7 inches; at flue, 2 feet;

TABLE XIV. — Details of Electrolytic Refineries.

	Great Falls.	Anaconda.
Number of tanks	312	1,400
Length " "	9 ft. 9 in.	8 ft. 2 in. } inside meas.
Width " "	2 ft. 4 in.	4 ft. 5 in. } inside meas.
Depth " "	3 ft. 9 in.	4 ft. 6 in. } inside meas.
Number of tanks per set, *i. e.*, number of tanks using same solution	204	200
Number of anodes per tank	20	38
" " cathodes per tank	20	80
" " amperes per square foot of cathode area	40	10
Power:		
Number of generators	2	7
" " circuits	1	7
Amperes generated per circuit	9,000	4,000
Volts per circuit	200	60
Character of material worked	Converter anodes	Casting furnace anodes
Kind of moulds used	Copper	Iron
Casting machines, pattern	Walker	Anaconda
Converter anodes, per cent. Cu	99.1	. . .
" " " " As and Sb	0.07	. . .
" " oz. Ag	56.00	. . .
" " oz. Au.	0.20	. . .
" " per cent. scrap	8.00	. . .
" " mud, per cent. Cu	41.0	. . .
" " " " oz. Ag	7,000.0	. . .
" " " " oz. Au	18.0	. . .
Refined anodes, per cent. Cu	99.27	99.25
" " " " As and Sb	0.07	0.10
" " oz. Ag	61.14	90.00
" " oz. Au	0.22	0.50
" " per cent. scrap	5.5	7.00
" " mud, per cent. Cu	18.00	10.00
" " " " oz. Ag	15,000.0	18,000.0
" " " " oz. Au	38.00	100.00
Size of anodes:		
Length	35 in.	$32\frac{7}{8}$ in.
Width	24 in.	$24\frac{1}{2}$ in.
Thickness	3 in., $2\frac{1}{2}$ in.	$1\frac{1}{4}$ in., 1 in.
Weight in pounds, converter	500	. . .
" " " refined	632	230
Number of days corroding, refined anode	. . .	37
" " " " converter anode	18	. . .
Size of cathodes:		
Length	$36\frac{1}{2}$ in.	33 in.
Width	26 in.	11 in.
Weight of starting sheets, lbs.	$2\frac{1}{2}$	$\frac{3}{4}$, 1
Weight of cathodes, lbs.	55	100
Number of cathodes on one bar	1	4
Electrolyte:		
Sulphuric acid per litre, grms.	170	150
Copper per litre, grms.	42	40
Temperature at head tank, C.°	64	50
Circulation per minute, gals.	6	3
Difference in potential:		
Soluble anodes, per tank	0.6	0.3
Insoluble " " "	2.5	2.0
Composition of cathodes, per cent. Cu	99.95	99.96
" " " " " As	0.0012	0.0009
" " " " " Sb	0.0033	0.0023
" " " oz. Ag	1.0	0.25

depth of metal, 12 to 13 inches; length of grate, 8 feet 6 inches; width, 7 feet 1.5 inches; depth of grate below top of bridge, 2 feet 6 inches; ratio of grate to hearth area, 1 : 3.7; height of roof above bridge, 1 foot 10 inches; above hearth at bridge, 2 feet 3 inches; at flue, 1 foot 10 inches; rise of arch, 1 inch per foot; size of flue at vulcatory, 3 feet 9 inches by 19 inches; of flue leading to chimney, 24 by 19 inches; size of chimney, 32 by 32 inches; height, 80 feet. Charge: liquid metal, 125,000 pounds; cathodes, 75,000 pounds. Analysis of bituminous coal: ash, 18.8; fixed carbon, 54.6; proportion of charge to coal by weight, 3 : 1; tons of charge to number of poles, 4 : 1; tons of charge to bushels of charcoal, 1 : 1.

The method of working differs little from that in common use, excepting that flapping or rabbling is replaced by blowing with compressed air (see *ante*); the metal bath in poling is covered with charcoal only toward the end of the operation, a few old poles, however, floating on the metal; of the two poles inserted through the working door, one is depressed near the door, the other toward the middle of the furnace; ingots, cakes, and wire bars are cast by means of suspended ladles 14 inches and 19 inches in diameter. All moulds are of copper. Cake and wire-bar moulds are painted with a mixture of lamp black and benzine; they are warmed at first by pouring in some copper. Ingot moulds are smoked before the first round by means of burning rosin held in a ladle. The tests applied to the copper are those of the Bell Telephone Company for Number 12 wire: conductivity, 96 to 97 per cent.; tensile strength, 65,600 pounds; torsion, forty twists in 6 inches between the clamps with 1 per cent. elongation. Three test bars are made from each charge, one bar being made when dipping begins, one when the charge is half out, and the third at the conclusion of the ladling. The test bars are 8 inches long, and have the full cross section of a regular wire bar. From each of these bars a section 6 inches long by 1 inch square is sawed; this section is rolled into a rod and afterward drawn cold to a Number 12 wire, Brown and Sharpe gauge, on which all tests are made. Conductivity tests are made with the Willyoung conductivity bridge.[1]

In conclusion, the writer wishes to express his thanks to the management and officers of the different companies for the courtesies shown him, and for the privilege of presenting to the Institute that part of his notes which is of general metallurgical interest.

[1] M. E. Leeds & Co., Philadelphia, Pa.

Appendix.

TABLE XV. — The Efficiency of Some Rocky Mountain Coals.*

	Summary of Chemical Tests.							Summary of Efficiencies.					
	Volatile matter.	Fixed carbon.	Total.	H_2O.	Ash.	Calories.	B. T. U.	Price per ton.	Evaporation from and at 212° F.	Cost to evaporate 1,000 lbs. water.	Relative value.	B. T. U. of coal.	Boiler efficiency.
Trail Creek (Mt. House) nut	25.92	48.85	74.77	9.04	16.19	7,534	13,561	$2.85	4.65	$0.306	$2.68	10,141	44.27
Trail Creek (Mt. House) lump	25.34	52.62	77.96	8.75	13.29	7,734	13,921	4.15	5.70	0.360	3.28	10,863	50.00
Red Lodge lump	26.24	52.39	78.63	9.23	12.14	7,245	13,041	4.50	6.70	0.345	3.74	10,255	61.00
Bridger lump	26.55	54.00	80.55	7.67	11.78	7,465	13,437	4.50	6.10	0.371	3.51	10,824	54.30
Sheridan, Wyoming, lump	26.78	46.40	73.18	21.20	5.62	7,006	12,611	5.50	5.01	0.547	2.88	9,229	52.40
Red Lodge washed nut	26.15	53.52	79.67	9.05	11.28	7,205	12,970	3.75	5.40	0.346	3.11	10,332	50.60
Chestnut washed mine run	23.10	50.18	73.28	8.64	18.08	8,407	15,133	3.05	5.30	0.288	3.05	11,090	46.50
Gebo lump	21.46	47.40	68.86	6.42	24.72	8,241	14,834	4.00	5.75	0.346	3.31	10,215	54.40
Bridger nut	30.00	50.64	80.64	3.95	15.41	7,195	12,952	3.75	4.86	0.386	2.80	10,444	44.80
Trail Creek (Kountz's) lump	26.24	47.65	73.89	12.41	13.70	7,167	12,901	4.50	5.44	0.413	3.13	9,533	55.00
Belt washed nut †	22.65	55.00	77.65	4.23	18.12	7,445	13,402	4.50	5.60	0.400	3.22	10,406	52.00
Galt (Canada) lump †	24.28	57.45	81.73	8.00	10.27	7,421	13,359	6.50	6.05	0.535	3.49	10,019	53.10
Rock Springs (Wyoming) lump †	27.90	61.31	89.21	5.64	5.15	7,686	13,836	6.75	7.55	0.447	4.35	12,343	59.00
Mountain Side mine run	26.84	49.91	76.75	2.71	20.54	7,918	14,252	3.40	5.78	0.294	3.32	10,938	51.00

*Taken from a paper read before Montana Society of Engineers by Prof. W. H. Williams, State Agricultural College, Bozeman, Montana, through *The Western Mining World*, Nov. 15, 1902.

† Price in Butte. Relative value compared to chestnut-washed mine run.

RESULTS OF TESTS MADE IN ENGINEERING LABORATORIES.

XIV.

STEAM.

TEST OF THE WESTINGHOUSE UNIT AT LINCOLN WHARF POWER STATION OF BOSTON ELEVATED RAILWAY COMPANY, MAY 7, 8, 9, 1903.

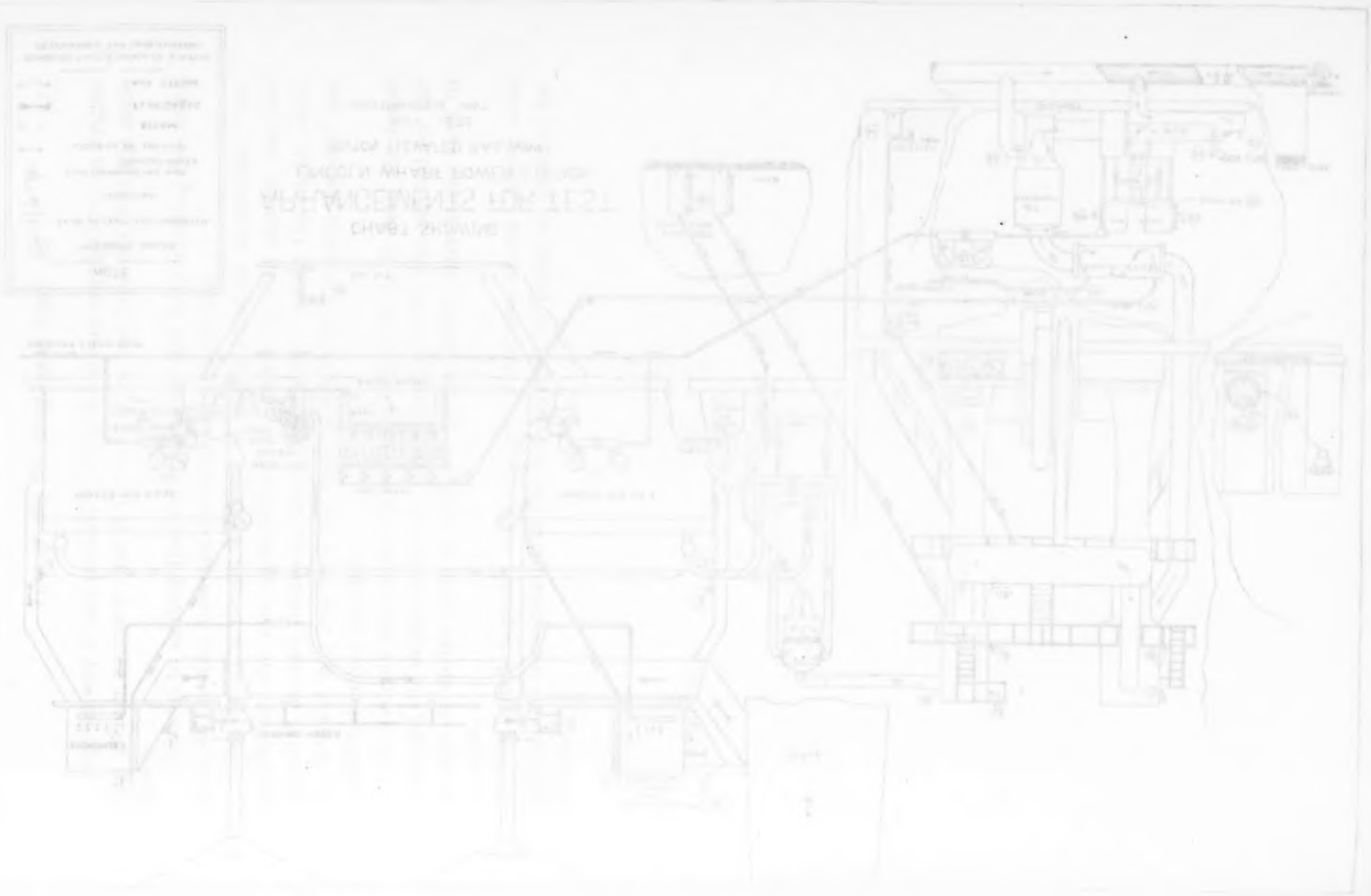
THROUGH the courtesy of the directors of the Boston Elevated Railway Company and Mr. C. F. Baker, Master Mechanic, permission to test the Lincoln Wharf plant was given the senior class of engineering students of the Massachusetts Institute of Technology.

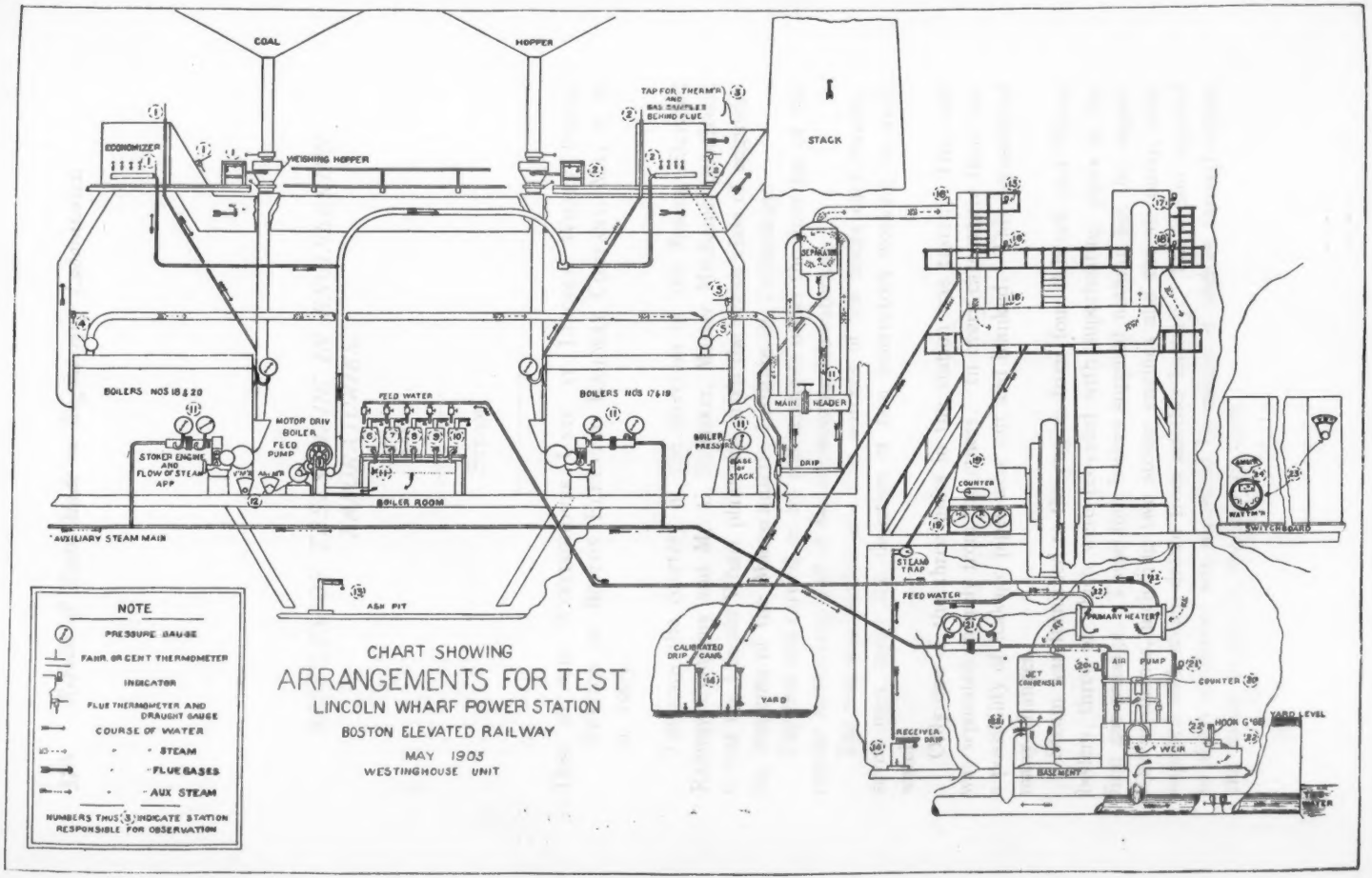
The test was carried on by 108 students under the direction of the regular instructing staff of the engineering laboratories.

The test was divided into four watches of ten hours each, twenty-seven men under the direction of two instructors working on each watch.

Only one of the three units in the station was tested. This unit was separated entirely from the others. In every case where there was a possibility of leakage pipes were cut and blanked. This necessitated many changes.

Steam was furnished to the engine from four Babcock and Wilcox boilers, three of which were provided with superheating pipes in the first gas passage. These four boilers supplied nothing but the engine. The auxiliaries, including two stoker engines and the air pump, were supplied with steam drawn from another battery. The steam supplied to these auxiliaries was measured by means of orifices through which the steam flowed on the way to each.





NOTE

- PRESSURE GAUGE
- FAHR. OR CENT. THERMOMETER
- INDICATOR
- FLUE THERMOMETER AND DRAUGHT GAUGE
- COURSE OF WATER
- STEAM
- FLUE GASES
- AUX. STEAM

NUMBERS THUS (3) INDICATE STATION RESPONSIBLE FOR OBSERVATION

CHART SHOWING

ARRANGEMENTS FOR TEST

LINCOLN WHARF POWER STATION

BOSTON ELEVATED RAILWAY

MAY 1903

WESTINGHOUSE UNIT

In this way it was possible to find the steam supplied to the engine with greater accuracy than if the auxiliaries had been fed from the same boilers.

Previous to starting the test the engine had been running for some hours at the load maintained constant during the test.

The electrical output was kept as nearly constant as possible; during the day this was accomplished by making up with another machine the extra amount demanded from this station; during the night, when the demand was less than the load on the engine, a water rheostat was connected across the line.

As students were stationed at five different levels in both the boiler room and the engine room, the accompanying chart was printed for the students to enable them to locate all points where observations were to be made.

This chart together with the list of stations shows clearly what observations were taken and the arrangement of piping as changed for the test.

LIST OF STATIONS, DUTIES, ETC. LINCOLN WHARF POWER STATION, ATLANTIC AVENUE, BOSTON.

No. of Station.	Duties.	Location.
1	Weigh coal for boiler No. 18 Weigh coal for boiler No. 20 Take temperature in flue, boilers No. 18 and No. 20 Take draft readings in flue, boilers No. 18 and No. 20 Take temperature of feed water entering and leaving economizer	Economizer level, boilers No. 18 and No. 20
2	For boilers No. 17 and No. 19 Same as Station 1	Economizer level, boilers No. 17 and No. 19
3	Take temperature in main flue Take samples and analyze gases from main flue	Economizer level, boilers No. 17 and No. 19
4	Take temperature of steam from boiler No. 18 Take temperature of steam from boiler No. 20 Take draft in uptake, boiler No. 18 Take draft in uptake, boiler No. 20 Take temperature in uptake, boiler No. 18 Take temperature in uptake, boiler No. 20	Top of boilers No. 18 and No. 20
5	Same as Station 4 for boilers No. 17 and 19	Top of boilers No. 17 and No. 19
6	Weigh feed water	Boiler room. Water weighing platform
7	Weigh feed water	
8	Weigh feed water	
9	Weigh feed water	
10	Weigh feed water	
11	Read boiler gauge at header Take temperature of steam at header Take pressures at flow of steam apparatus for stoker engine, boilers No. 17 and No. 19 Take pressures at flow of steam apparatus for stoker engine, boilers No. 18 and No. 20 Take temperature of feed water in tank	Boiler room floor
12	Take ammeter and voltmeter readings for feed-pump motor	Boiler room floor
13	Weigh ashes, boilers No. 18 and No. 20 Weigh ashes, boilers No. 17 and No. 19	Ash pit floor
14	Take drip-can readings for separator Take drip-can readings for throttle and valve chest Weigh drip barrel from receiver drip trap	Yard near boiler house
15	Take indicator cards: high pressure cylinder, head end Mark cards "H. H.," also with date and time	Engine platform
16	Take indicator cards: high pressure cylinder, crank end Mark cards "H. C.," also with date and time Take temperature of steam supply Take temperature of steam in receiver	Engine platform
17	Take indicator cards: low pressure cylinder, head end Mark cards "L. H.," also with date and time	Engine platform
18	Take indicator cards: low pressure cylinder, crank end Mark cards "L. C.," also with date and time Collect all four cards and carry to planimeter table	Engine platform
19	Keep time Ring gong every fifteen minutes. One stroke on the minute Two strokes ten seconds before minute Three strokes thirty seconds before minute Read pressures at gauge board Boiler, receiver, vacuum Read engine counter and assist at planimeter table on air-pump cards	Engine room floor
20	Read counter at air pump. Stroke Take indicator cards: air pump, head end, and pump end. High pressure cylinder Mark cards "A. H.," also with date and time Carry all cards to planimeter table	Engine foundation floor

LIST OF STATIONS, DUTIES, ETC. LINCOLN WHARF POWER STATION, ATLANTIC AVENUE, BOSTON. — *Continued.*

No. of Station.	Duties.	Location.
21	Take indicator cards: air pump, head end and crank end. Low pressure cylinder Mark cards "A. L.," also with date and time Take temperatures and pressures flow of steam apparatus	Engine foundation floor
22	Read hook gauge at weir Take temperature of feed water entering and leaving heater Take temperature of injector water entering and leaving condenser	Engine foundation floor
23 and 24	Read wattmeter and voltmeter, one-half minute intervals	Switch board
25	Planimeter areas. High pressure cylinder cards Draw and measure ordinates, length, per cents., cut-off, release, and compression Tabulate results	Engine room floor
26	Same as Station 25 for low pressure cylinder cards	Engine room floor
27	Planimeter areas: air pump cards Draw end ordinates and measure lengths Tabulate results	Engine room floor

REMARKS.

There was some difficulty in keeping the safety valves on the boilers from blowing. To prevent this, air was admitted to the boilers through the cleaning doors on the side of the setting. This cut down the efficiency of the boilers and also reduced the saving made in the economizers.

Samples of flue gas showed a large percentage of free oxygen and but little carbonic acid.

With three boilers supplying the steam instead of four, the economy of the plant would have been greater.

A considerable quantity of coal dropped through the grates into the ash pit. This coal was weighed back and returned to the coal pocket according to the regular custom at this station.

Twice during the test the engine exhausted outboard owing to the stopping of the air pump. The total time during which the engine was without vacuum was a little less than two hours.

The electrical output of the engine was dropped about 50 per cent. during these periods.

The thermometers used in measuring the temperature of the steam at the top of the boilers gave considerable trouble from the separation of the mercury thread due to vibrations. This may have caused an error of from one to two degrees in the average of some of these readings.

On calculating the tests it was found that the output of electrical horse power differed from the indicated power only by about 112 horse power.

The indicator springs were tested and found to be practically correct. The electrical instruments were calibrated a second time without appreciable change of result. It was finally discovered that there was some lost motion in the indicator rigs. The indicator rig consisted of a long rod which ran up the side of the cylinder, and which received a partial rotation at each stroke. This motion was obtained from the crosshead through a stud which bore against a helical plate fastened to the long rod. A spring served to keep the stud and helix in contact.

To determine the error due to this lost motion two diagrams were taken on the same card, one with the lost motion taken up and one with the indicator rig as it commonly runs.

The engine at the time these cards were taken was under a constant load, which was kept the same as during the test. From the various sets of cards taken it seems that the indicated power of the engine should be increased by about 4.5 per cent. to get the correct value.

TEST OF THE LINCOLN WHARF POWER STATION OF THE BOSTON ELEVATED RAILWAY COMPANY.

Date of test, Thursday, May 7, 1903, 2.15 P.M., to Saturday, May 9, 1903, 6.15 P.M.
Duration of test, forty hours.

Engine Room Data.

Main Engine:

Type of engine, Westinghouse-Corliss, vertical cross compound.	
Diameter of high pressure cylinder	44 inches
Diameter of high pressure piston rod	9 inches
Diameter of low pressure cylinder	87 inches
Diameter of low pressure piston rod	10 inches
Stroke	60 inches
Ratio of the volumes of the high and of the low pressure cylinders	1 to 3.91
Ratio of the crank to the connecting rod	1 to $5\frac{1}{2}$
Revolutions per minute	74.37

Pressures and Temperatures:

Pressure of steam at throttle (gauge)	158.1 pounds
Temperature of steam at throttle	439.3° F.
Temperature corresponding to throttle pressure absolute	369.6° F.
Degrees of superheat	69.7° F.
Pressure in receiver (gauge)	16.9 pounds
Temperature of steam in the receiver	301.5° F.
Vacuum in condenser (gauge)	11.8 pounds
Barometer	14.7 pounds

Events of Stroke. Per Cent. of Piston Displacement:

	High.			Low.	
	Head end.	Crank end.		Head end.	Crank end.
Cut-off . .	25.6	25.5	Cut-off . .	38.2	32.8
Release . .	98.2	97.0	Release . .	95.3	95.6
Compression .	8.4	13.5	Compression .	. .	. .

Pressures from Indicator Cards (Absolute, Pounds per Square Inch):

High Pressure Cylinder.			Low Pressure Cylinder.		
	Head end.	Crank end.		Head end.	Crank end.
Initial . . .	165.4	168.5	Initial . . .	31.3	30.6
Cut-off . .	136.6	135.6	Cut-off . .	23.2	24.1
Release . .	40.1	44.7	Release . .	9.4	9.8
Compression .	39.2	41.7	Compression .	5.4	4.9
M. E. P. . .	55.34	57.66	M. E. P. . .	14.14	13.49

Air Pump:

Type, Blake Vertical Compound, with link cross connection between piston rods	16″ and 28″ by 21″ stroke
Revolutions of pump per minute	23.75

Flow of Steam for Air Pump:

Diameter of orifice used	.636 inches
Pressure on supply side of orifice (gauge)	149.7 pounds
Temperature of steam on supply side of orifice	371.5° F.
Pressure on delivery side of orifice (gauge)	90.0 pounds
Temperature of steam on delivery side of orifice	349.7° F.

Condensing Water and Condensed Steam passing over Weir:

Hook gauge (depth of water passing over weir)	8.78 inches
Width of weir	2 feet
Distance from crest of weir to bottom of weir box	1 foot
Temperature of water entering condenser	50.5° F.
Temperature of water and condensed steam passing over weir . .	93° F.

Formula for calculating cubic feet per second of water passing over weir:

$$Q = 3.36\, b \left[(H + h)^{\frac{3}{2}} - h^{\frac{3}{2}} \right].$$

H = hook gauge head in feet.
h = head due to velocity of approach.
b = length of crest in feet.

Drip Steam:

Total drip from separator (forty hours)	1,650 pounds
Total drip from throttle, valve chest, etc. (forty hours) . . .	1,610 pounds
Total drip from receiver (forty hours)	920 pounds

Boiler Room Data.

Coal:

Total coal weighed in hoppers (forty hours)	315,370 pounds
Total coal dust weighed back and returned to the coal pocket . .	56,948 pounds
Total coal burned (forty hours) charged to the boilers . . .	258,422 pounds
Total ash (forty hours)	50,776 pounds

Totals for Each Boiler:

(*a*) Total coal weighed in hoppers (forty hours), pounds.
(*b*) Total coal dust weighed back and returned to coal pocket (forty hours), pounds.
(*c*) Total ash (forty hours), pounds.

	Boiler No. 17.	Boiler No. 19.	Boiler No. 18.	Boiler No. 20.
(*a*)	94,460	82,856	67,888	70,166
(*b*)	24,577	19,350	6,436	6,585
(*c*)	14,177	12,421	11,743	12,435

Chemical Analysis of Coal:

	Boiler No. 17.	Boiler No. 19.	Boiler No. 18.	Boiler No. 20.	Average of four.
Volatile matter, per cent.	20.63	19.38	21.88	19.02	20.23
Fixed carbon, per cent.	65.39	70.11	66.55	70.66	68.13
Ash, per cent.	11.71	9.00	9.62	8.72	9.76
Moisture, per cent.	2.27	1.52	1.85	1.61	1.81

Heat of combustion by Mahler bomb . . . . . 14,420 B. T. U.

Chemical Analysis of Ash — Average of Three Determinations:

Volatile matter, per cent.	14.67
Fixed carbon, per cent.	65.68
Ash, per cent.	19.65
Moisture, per cent.	0.00

Water and Steam:

Total feed water (forty hours)	1,889,325 pounds
Total steam supplied to main engine (forty hours)	1,886,065 pounds

Temperatures of the Feed Water:

Entering the primary heater	54.7° F.
Leaving the primary heater	135.5° F.
At tank on boiler room floor	133.2° F.
Entering economizer of boilers No. 17 and No. 19	133.5° F.
Leaving economizer of boilers No. 17 and No. 19	205.5° F.
Entering economizer of boilers No. 18 and No. 20	135.1° F.
Leaving economizer of boilers No. 18 and No. 20	217.9° F.

Temperature of Steam at Boilers:

No. 17 = 436.3° F. No. 19 = 471.4° F. No. 18 = 443.7° F. No. 20 = 367.7° F.

Boiler pressure at header (gauge)	161.6 pounds
Temperature of steam at header	450.9° F.
Degrees of superheat in steam at header	79.6° F.

Temperature and Draft of Flue Gases in Uptake:

	Boiler No. 17.	Boiler No. 19.	Boiler No. 18.	Boiler No. 20.
Temperature	400.8° F.	413.8° F.	456.8° F.	492.6° F.
Draft (inches of water)	.57	.59	.55	.55

Temperature of flue gas in main flue near stack . . . . . 282.0° F.

Temperature of flue gas in flue . . . { Boilers No. 17 and No. 19 = 289.4° F. / Boilers No. 18 and No. 20 = 291.4° F.

Flow of Steam (Stoker Engines):

	Boilers No. 17 and No. 19.	Boilers No. 18 and No. 20.
Diameter of orifice used (inches)	.204	.205
Pressure on supply side of orifice (gauge)	146.1 pounds	153.0 pounds
Pressure on delivery side of orifice (gauge)	130.0 pounds	138.2 pounds

Surfaces:

Total heating surface, four boilers (square feet)	21,200
Total grate surface, four boilers (square feet)	360

RESULTS OF ENGINE TEST.

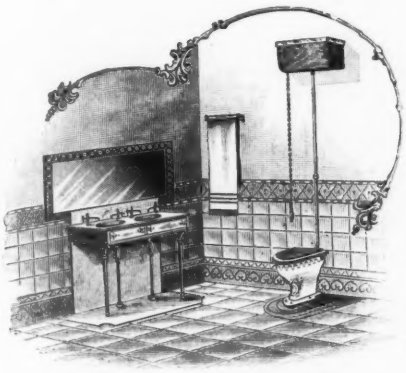
ENGINE, HORSE POWER, FROM CARDS AS TAKEN.	
High pressure cylinder Head end, 948.2; crank end, 946.6. Total	1,894.8 H. P.
Low pressure cylinder Head end, 947.2; crank end, 891.7. Total	1,838.9 H. P.
Total horse power of engine	3,734.0 H. P.
Horse power of engine corrected for error of indicator rig (see Remarks)	3,902.0 H. P.

	FROM DATA.	CORRECTED VALUES.
Steam furnished engine per horse power per hour	12.63 lbs.	12.08
B. T. U. per horse power per minute (actual vacuum)	235.7	223.4
Condensing water and condensed steam passing through weir per hour	1,007,223 lbs.	
Condensing water per hour	960,095 lbs.	
Condensing water per corrected horse power per hour	246.2 lbs.	
Thermal efficiency of engine	18.00%	
Efficiency of Carnot engine working between same temperatures	33.24%	
Thermal efficiency of engine in per cent. of Carnot engine efficiency	54.14%	
Electrical horse power output, 2,702 kilo watts	3,622 H. P.	
Mechanical efficiency of engine	92.8%	
Dry coal per electrical horse power output per hour	1.75 lbs.	
Dry coal per kilo watt per hour	2.34 lbs.	
Steam used by air pump (forty hours)	105,470 lbs.	
Steam furnished to air pump in per cent. of engine steam	5.59%	
Air pump steam in per cent. total steam to main engine and air pump	5.295	

The results of the boiler test are given on the next page.

RESULTS OF BOILER TEST.

Total water feed to boilers (forty hours)	1,889,325 lbs.
Total coal charged to boilers (forty hours)	258,422 lbs.
Per cent. moisture in coal	1.81
Total dry coal burned (forty hours)	253,745 lbs.
Degrees of superheat in steam at header	79.6° F.
Average temperature of feed water entering economizers	134.3° F.
Average temperature of feed water leaving economizer	211.7° F.
Total B. T. U. taken up by one pound of water (boilers and economizers)	1,130.8 B. T. U.
Equivalent evaporation from and at 212° F. per pound of dry coal	8.718 lbs.
Boiler horse power developed	1,603
Efficiency of boiler plant (per cent.)	58.4
Heat recovered by economizer in per cent. of total heat taken up from coal	6.9%
Dry coal burned per square foot of grate surface per hour	17.62 lbs.
Water evaporated from and at 212° F. per square foot of heating surface per hour	2.604 lbs.
Heat recovered from engine exhaust by primary heater per pound of feed water	80.9 B. T. U
Heat recovered from primary heater per pound of coal	591.4 B. T. U.
Per cent. of saving in coal effected by primary heater	7.15%
Steam used in forty hours by stoker engines	13,172 lbs.
Stoker engine steam in per cent. of engine steam	.698%
Steam for stoker engines in per cent. total steam to main engine and stokers	.697%
Steam for air pump and stoker engines in per cent. total steam for plant	5.91% (inc. drip), 5.92%



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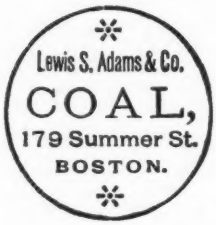

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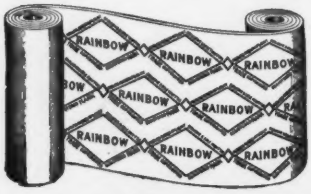

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